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METAL-MINING PRACTICE

BY

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METAL-MINING PRACTICE[1]

By CHAS. F. JACKSON[2] and J. H. HEDGES[3]

INTRODUCTION

This bulletin summarizes mining practices of a large number of metal mines in the United States and foreign countries. The adaptation of mining methods best suited to the various natural conditions in mines is discussed, and comparative costs are presented. The basic data—gathered in the field by Bureau of Mines engineers and consultants in cooperation with mine operators—have been published in a series of information circulars dealing with practices and costs at individual mines and ore-dressing plants. The data have been made comparable by adhering to a standard form of presentation in the individual reports.

The first circular was issued in December 1928, and by December 31, 1938, 250 dealing with mining and milling of metallic ores had been published. Of this number, 164 deal with operations at individual mines and plants—94 with mining only, 59 with milling only, and 11 with mining and milling. Eleven circulars are summaries, each covering a particular mining method, and 53 are discussions of separate operations involved in mine operation or deal with special phases of mining. Seven of them treat special ore-dressing and allied subjects, and 15 are a part of a mineral-industries survey by counties or districts, in which many individual properties are described and costs and other data are presented. Earlier bulletins summarized the mining and milling of ores of the principal metallic minerals,[4] or specific problems[5] in the exploitation of metal mines.

OBJECT AND SCOPE OF BULLETIN

The purpose of this bulletin is to consolidate and analyze the information contained in earlier reports on mining methods, practices, and costs.

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[1] Work on manuscript completed March 1939.

[2] Chief engineer, Mining Division, Bureau of Mines.

[3] Assistant to the Director, Bureau of Mines.

[4] Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bureau of Mines Bull. 306, 1929, 192 pp.

Jackson, Chas. F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, 151 pp.

Jackson, Chas. F., Knaebel, John B., and Wright, C. A., Lead and Zinc Mining and Milling in the United States; Current Practices and Costs: Bureau of Mines Bull. 381, 1935, 204 pp.

Gardner, E. D., Johnson, C. H., and Butler, B. S., Copper Mining in North America: Bureau of Mines Bull. 405, 1938, 300 pp.

[5] McElroy, G. E., Ventilation of the Large Copper Mines of Arizona: Bureau of Mines Bull. 330, 1931, 145 pp.

Jackson, C. F., and Knaebel, J. B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, 155 pp.

Gardner, E. D., and Johnson, J. F., Shaft-Sinking Practices and Costs: Bureau of Mines Bull. 357, 1932, 104 pp.

Jackson, Chas. F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, 296 pp.

Although the exploitation of deposits of metalliferous ores cannot be discussed comprehensively without considering milling and marketing of the ores, this bulletin is concerned principally with mining or ore production. In the following pages milling and marketing are discussed briefly only when they have a direct bearing on some phase of mine exploitation.

The circulars covering methods and practices at individual mines were written for the most part by members of the operating staffs in accordance with an outline prepared by the engineers of the Bureau of Mines. The outline was designed to produce papers that would (1) describe the characteristics of the ore deposits that governed the selection and application of the mining methods employed in each case, (2) describe the methods and practices employed, and (3) present the costs of mining under the described conditions by the methods used, thus giving a complete picture. The outline was responsible for uniform presentation of comparable data which it was possible to summarize and analyze in a series of reports on different phases of mining practice.

ACKNOWLEDGMENTS

The study of mining methods and costs covered by the circulars and bulletins mentioned above would have been impossible without the cooperation of the management and staff of the many companies that contributed information and of the executives of these companies who permitted the gathering and publication of the information.

The authors acknowledge this cooperation without specifically mentioning names. Original authors, both of the information circulars and of articles in the technical press from which data have been taken, are given in footnote references in the body of the bulletin. If any have been omitted, the omission is unintentional. These references not only serve as acknowledgments but also assist the reader in running down details impossible to include within the scope of the present summary.

DEFINITIONS

The terminology used in this bulletin is believed to be that employed and understood by the average American metal miner, shift boss, foreman, superintendent, engineer, or manager, to whom the bulletin probably will be of most interest. This may vary somewhat from dictionary definitions; hence, to avoid possible misunderstanding, a number of terms are defined as they are employed in this paper

MINE.—A pit or excavation in the earth from which ores or other mineral substances are taken by digging; hence, either an open-pit or an underground working.

PROSPECTING.—The search for outcrops or surface exposure of mineral deposits.

EXPLORATION.—The search for unknown ore deposits or the extension of known deposits, including preliminary development and tests to determine probable value. Exploration may be conducted from the surface or from underground workings, by surface trenching, core or churn drilling, or by driving drifts, crosscuts, raises, and shafts. The term is used here to cover more comprehensive work than prospecting, and includes sampling and other procedure for obtaining information upon which to base ore-reserve estimates or plans for future mining operations.

DEVELOPMENT.—The preparation of a mine for ore extraction, including construction of all openings required for ventilation, drainage, and transportation of broken ore to the surface, such as shafts, tunnels, main raises, crosscuts, and haulage drifts, skip pockets, and pump stations.

Stope development.—The driving of subsidiary openings designed to prepare blocks of ore for actual extraction by stoping.

Stoping.—The act of excavating ore by means of horizontal, vertical, or inclined workings in veins or large irregular bodies of ore, or by rooms in flat tabular deposits, and also the mining of ore by caving methods. It covers the breaking and removal of ore from underground openings except those driven for exploration and development. In many of the information circulars upon which this bulletin is based the term "mining" was used synonymously with "stoping" and is often so used in practice.

Underground transportation.—The transporting of ore, rock, men, materials, and supplies through shafts and haulageways, including the loading of ore or rock into cars and carrying it to the surface.

Mucking.—Hand or mechanical shoveling and power scraping of ore and rock. Because the term covers both shoveling and power scraping and because it has become thoroughly established in mining parlance through universal use, it is employed freely here in this sense, even though its dictionary meaning is something quite different.

PROSPECTING AND EXPLORATION

In this presentation of the technology of mine exploitation, the various operations will be discussed in the order in which they are ordinarily performed in the discovery and exploitation of an actual mining property. Thus, the search for valuable mineral by the prospector will be discussed first.

In North America the initial discoveries of valuable metal mines in a district have seldom, if ever, been the result of organized exploration of large areas by well-financed companies. Prospecting for metallic ores has not usually been conducted as has prospecting for petroleum—by test-drilling or "wild-catting" in areas indicated by structural data (often gained from elaborate geological and geophysical surveys) to be favorable for the accumulation of oil. Certain noteworthy exceptions are found in the exploration for iron ore in the Lake Superior region and for lead and zinc ores in the Southeastern Missouri and Tri-State districts. Even in these districts, however, the original discoveries of ore minerals doubtless were made by chance and at the outset large-scale exploration was not carried on.

It is true that large areas of North America have been mapped and studied carefully by Government geologists, and the published results of their surveys have been very useful in the search for ore. However, particularly in the United States, these valuable reports have not been used by the average prospector as freely and intelligently as they should have been.

With the depletion of known and more easily discovered ore deposits and the expected advance in geologic knowledge and application of geophysical methods, prospecting for ore may some day be conducted on a scale and in a manner analogous to petroleum exploration, covering wide areas that the geologist has designated as favorable for ore deposition. In the past, however, most original discoveries of ore in this country have been the result either of accident or of the persistent efforts of individuals or small groups of prospectors, often ill-equipped and poorly financed.

PROSPECTING

In the early days, the search for ore often was conducted in remote regions, far from any base of supplies. Equipment and food had to be packed on foot or on the backs of animals, often for long distances.

The prospector was forced to provide his own shelter and prepare his own food, perhaps to procure part of his food supply with rod and gun, and to perform many tasks that absorbed much of the time and energy that might otherwise have been spent in the actual search for ore.

The life of the old-time prospector was an arduous and usually a lonely one, and only those with natural physical endurance beyond the ordinary, together with determination and extraordinary optimism, were fitted for it. Relatively few prospectors have received financial rewards commensurate with the efforts put forth and the privations suffered; where they have persisted for years without success it has probably been due to particular aptitude, a liking for the life, and the undying hope of some day making a big "strike."

Today, supply bases are closer and more easily accessible in this country, due to railroad, highway, and branch-road construction. In many remote regions in North America and other continents modern means of transportation, notably airplanes and motorized boats, have brought prospecting fields immeasurably nearer, in point of time and effort, to bases of supply.

Prospecting still requires special natural adaptability and, now more than ever, a knowledge of the different ore minerals, the geological conditions favorable for ore deposition, and the different rocks and rock structures friendly to mineralization.

With the revival of interest in gold mining brought about by the depression of the nineteen thirties and the consequent resumption of prospecting for gold, many classes for prospectors were established by Government agencies, schools, and colleges throughout the world. This was a step in the right direction and doubtless has resulted in more intelligent prospecting than would otherwise be done, although classroom instruction alone cannot engender the qualties so conspicuously displayed by the early prospectors. Through long experience and observation the old-timer often amassed a fund of information concerning the various minerals, their mode of occurrence, and the rocks with which they are associated that gave rise to that intuitive phenomenon popularly called "a nose for ore."

EQUIPMENT

The equipment required varies with the type of prospecting to be done, the climatic and physical characteristics of the country, mode of transportation, length of the campaign, and the funds available. Gardner [6] has discussed and listed the usual requirements in the western United States for transportation, camps, tools, cooking, provisions, clothing, and first-aid supplies. For the purpose of the present discussion, it is sufficient to note that all these different types of equipment and supplies must be considered.

MINING LAWS

Knowledge of Federal and State or Provincial laws relating to locating, staking, and recording claims on public lands in the countries and areas in which the prospector plans to work is essential.

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[6] Gardner, E. D., and Johnson, Fred W., Prospecting for Lode Gold and Locating Claims on the Public Domain: Bureau of Mines Inf. Circ. 6843 (revised), 1935, pp. 8–10.

Obviously, prospecting on private lands can only be done with the consent of and by arrangement with the owner. The reader is referred to Bulletin 94 for a discussion of the United States mining laws.[7] Reference is also made to Circular 1278, United States Department of the Interior, General Land Office. A summary of these laws as they apply to location of lode claims is given in Information Circular 6843 (revised).[8] A summary of the placer-mining laws is given in Information Circular 6611R.[9]

The Bureau of Mines has issued a number of circulars summarizing the mining laws of other countries.[10]

PLACER DEPOSITS

Prospecting for placer gold, except perhaps in the case of buried placers, is the simplest form of prospecting.

Gold, platinum, and tin are the principal metallic minerals won from placers, but gold (alloyed with varying percentages of silver) is the only metal that has been recovered in commercially important quantities from placers in the United States. The following brief discussion applies to gold placers.

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[7] Thompson, J. W., United States Mining Statutes Annotated: Bureau of Mines Bull. 94, 1915, 1,772 pp.

[8] Gardner, E. D., and Johnson, Fred W., Prospecting for Lode Gold and Locating Claims on the Public Domain: Bureau of Mines Inf. Circ. 6843 (revised), 1935, pp. 11–18.

[9] Johnson, Fred W., and Jackson, Chas. F., Federal Placer-Mining Laws and Regulations, and Small-Scale Placer-Mining Methods: Bureau of Mines Inf. Circ. 6611R, 1938, 37 pp.

[10] Garman, A. D., Mining Laws of Argentina: Bureau of Mines Inf. Circ. 6251, 1930, 11 pp.
Frey, J. W., Mining Laws of Australia: Bureau of Mines Inf. Circ. 6334, 1930, 13 pp.
Youngman, E. P., Mining Laws of Austria: Bureau of Mines Inf. Circ. 6535, 1931, 12 pp.
——— Mining Laws of Belgium: Bureau of Mines Inf. Circ. 6278, 1930, 8 pp.
Garman, A. D., Mining Laws of Bolivia: Bureau of Mines Inf. Circ. 6140, 1929, 16 pp.
——— Mining Laws of Brazil: Bureau of Mines Inf. Circ. 6298, 1930, 10 pp.
Frey, J. W., Mining Laws of British Africa: Bureau of Mines Inf. Circ. 6105, 1929, 17 pp.
——— Mining Laws of British India: Bureau of Mines Inf. Circ. 6104, 1929, 3 pp.
——— Mining Laws of Canada: Bureau of Mines Inf. Circ. 6346, 1930, 45 pp.
Garman, A. D., Mining Laws of Chile: Bureau of Mines Inf. Circ. 6252, 1930, 6 pp.
Frey, J. W., Mining Laws of China: Bureau of Mines Inf. Circ. 6111, 1929, 6 pp.
Garman, A. D., Mining Laws of Colombia: Bureau of Mines Inf. Circ. 6131, 1929, 7 pp.
Aitkens, I., Mining Laws of Cuba: Bureau of Mines Inf. Circ. 6458, 1931, 11 pp.
Frey, J. W., Mining Laws of Czechoslovakia: Bureau of Mines Inf. Circ. 6103, 1929, 3 pp.
Youngman, E. P., Mining Laws of Denmark and Danish Possessions: Bureau of Mines Inf. Circ. 6631, 1932, 6 pp.
Aitkens, Irene, Mining Laws of the Dominican Republic: Bureau of Mines Inf. Circ. 6444, 1931, 8 pp.
Garman, A. D., Mining Laws of Ecuador: Bureau of Mines Inf. Circ. 6272, 1930, 7 pp.
Youngman, E. P., Mining Laws of Egypt: Bureau of Mines Inf. Circ. 6552, 1932, 16 pp.
——— Mining Laws of Ethiopia (Abyssinia): Bureau of Mines Inf. Circ. 6536, 1931, 15 pp.
——— Mining Laws of French Morocco: Bureau of Mines Inf. Circ. 6266, 1930, 11 pp.
——— Mining Laws of Greece: Bureau of Mines Inf. Circ. 6634, 1932, 9 pp.
Garman, A. D., Mining Laws of Guatemala: Bureau of Mines Inf. Circ. 6184, 1929, 14 pp.
Youngman, E. P., Mining Laws of Haiti: Bureau of Mines Inf. Circ. 6632, 1932, 10 pp.
Garman, A. D., Mining Laws of Honduras: Bureau of Mines Inf. Circ. 6214, 1929, 10 pp.
Youngman, E. P., Mining Laws of Hungary: Bureau of Mines Inf. Circ. 6450, 1931, 7 pp.
——— Mining Laws of Italy: Bureau of Mines Inf. Circ. 6363, 1930, 3 pp.
Garman, A. D., Mining Laws of Japan: Bureau of Mines Inf. Circ. 6297, 1930, 9 pp.
Youngman, E. P., Mining Laws of Latvia: Bureau of Mines Inf. Circ. 6542, 1932, 5 pp.
——— Mining Laws of the Republic of Liberia: Bureau of Mines Inf. Circ. 6630, 1932, 11 pp.
Frey, J. W., Mining Laws of the Federated Malay States: Bureau of Mines Inf. Circ. 6181, 1929 5 pp.
Lineberger, P. M., Mining Laws of Mexico: Bureau of Mines Inf. Circ. 6182, 1929, 14 pp.
Youngman, E. P., Mining Laws of the Netherland East Indies: Bureau of Mines Inf. Circ. 6451, 1931, 20 pp.
Frey, J. W., Mining Laws of New Zealand: Bureau of Mines Inf. Circ. 6185, 1929, 4 pp.
Garman, A. D., Mining Laws of Nicaragua: Bureau of Mines Inf. Circ. 6302, 1930, 7 pp.
——— Mining Laws of Paraguay: Bureau of Mines Inf. Circ. 6207, 1929, 5 pp.
Frey, J. W., Mining Laws of Persia: Bureau of Mines Inf. Circ. 6192, 1929, 2 pp.
Garman, A. D., Mining Laws of Peru: Bureau of Mines Inf. Circ. 6216, 1929, 17 pp.
Youngman, E. P., Mining Laws of Poland: Bureau of Mines Inf. Circ. 6629, 1932, 7 pp.
Frey, J. W., Mining Laws of Portugal: Bureau of Mines Inf. Circ. 6197, 1929, 5 pp.
Youngman, E. P., Mining Laws of Rumania: Bureau of Mines Inf. Circ. 6213, 1929, 18 pp.
Garman, A. D., Mining Laws of Salvador: Bureau of Mines Inf. Circ. 6336, 1930, 10 pp.
Youngman, E. P., Mining Laws of Spain: Bureau of Mines Inf. Circ. 6219, 1930, 16 pp.
Frey, J. W., Mining Laws of Trinidad: Bureau of Mines Inf. Circ. 6332, 1930, 3 pp.
——— Mining Laws of Turkey: Bureau of Mines Inf. Circ. 6188, 1929, 1 p.
Garman, A. D., Mining Laws of Uruguay: Bureau of Mines Inf. Circ. 6338, 1930, 7 pp.
——— Mining Laws of Venezuela: Bureau of Mines Inf. Circ. 6210, 1929, 10 pp.
Tyler, P. M., Mining Laws of Yugoslavia: Bureau of Mines Inf. Circ. 6183, 1929, 6 pp.

The most important type of placer deposit results from the weathering and disintegration of lodes and of rock formations containing gold disseminated through them, the erosion and transportation of these rocks by running water accompanied by further disintegration and liberation of the gold from the enclosing rock minerals, and the concentration of the gold in places where the velocity of the transporting medium (flowing water) is such that the gold and other heavy minerals come to rest but the lighter and finer material is carried away.

Another type of placer is the residual placer, consisting of disintegrated gold-bearing lode material in or very near its original position with only a relatively small amount of barren material removed by erosion; in some of these a considerable part of the soluble constituents may have been removed by chemical action, resulting in enrichment by gold.

Transported placers have been classified, upon the basis of their present position relative to that of streams and other waters, by Brooks [11] as follows:

Creek placers, beach placers, hillside placers, river-bar placers, gravel-plain placers, sea-beach placers, ancient beach placers, and lake-bed placers. The positions of the most productive types are shown in figure 1.

Placers consist for the most part of unconsolidated alluvium, rounded pebbles, and boulders; from the above brief discussion of their origin it is apparent that prospecting for them should begin along stream beds (existing or ancient) and bars, from which they may extend to benches and hillsides.

Ancient placers sometimes have been buried under considerable thicknesses of later glacial or stream deposits or lava flows; in such instances their discovery is more difficult.

For more comprehensive discussions of prospecting for placer gold the reader is referred to a few of the more recent [12] publications dealing with this subject; only the most important principles are summarized here.

Since placers are the result of transportation and concentration of gold particles by running waters, prospecting begins along streams. The most favorable streams are those originating in and flowing through areas of igneous rocks, which are the chief source of lode gold. If these rocks are known to contain gold the chances for the formation of placers along streams flowing through them obviously are enhanced. In actual prospecting, it more often happens that gold is found first in placers and the lode source of the gold is then sought, rather than the reverse. It should be noted, however, that in many instances valuable placers have been found and worked without finding the source of their gold, and many rich gold lodes have been worked in areas where no placers result from their erosion.

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[11] Brooks, Alfred H., The Mineral Deposits of Alaska. Mineral Resources of Alaska, 1913: Geol. Survey Bull. 592, 1914, pp. 25–32.

[12] Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bureau of Mines Bull. 259, 1927, pp. 31–40.

Johnson, Fred W., and Jackson, Chas. F., Placer-Mining Laws and Regulations and Small-Scale Placer-Mining Methods: Bureau of Mines Inf. Circ. 6611R, 1938, 37 pp.

Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. I. General Information, Hand-Shoveling, and Ground-Sluicing: Bureau of Mines Inf. Circ. 6786, 1934, pp. 23–46.

Vanderburg, William O., Placer Mining in Nevada: Nevada State Bureau of Mines and Mackay School of Mines, Bull. 4, vol. 30, May 15, 1936, pp. 29–34.

Ingersoll, Guy E., Hand Methods of Placer Mining and Placer-Mining Districts of Washington and Oregon; State College of Washington, Monthly Bull., vol. 14, No. 10, March 1932, 47 pp.

Staley, W. W., Elementary Methods of Placer Mining: Idaho Bureau of Mines and Geology, Pamphlet 35, May 1931, 20 pp.

von Bernewitz, M. W., Handbook for Prospectors: McGraw-Hill Book Co., New York, 3d ed., 1935, 372 pp.

Wilson, Eldred D., and Tenney, J. B., Arizona Gold Placers and Placering: Univ. of Arizona Bull., vol. 3, No. 1, Jan. 1, 1932, pp. 9–72.

Dingman, Oscar A., Placer-Mining Possibilities in Montana: Montana Bureau of Mines and Geology, Mem. 5, June 1932, 33 pp.

Placer gold, then, is usually first and most easily found along streams and their bars, even though the richest deposits may lie in benches or on hillsides above. Since the heaviest particles of gold are deposited nearest their source in the host rock, prospectors should work upstream from the original discovery, especially if the gold is fine. At some point upstream the gold may disappear, and it is then apparent that the source of the gold has been passed.

In prospecting along a stream, the prospector pans the gravel at various points, selecting particularly places that show concentrations of heavy minerals ("black sands") with which placer gold is commonly associated. These usually will be found on and just above bedrock and in cracks and crevices therein. Depressions in the bedrock are favorable places for concentration. Excavating may be necessary to reach bedrock in the stream bed and its bars and along its banks.

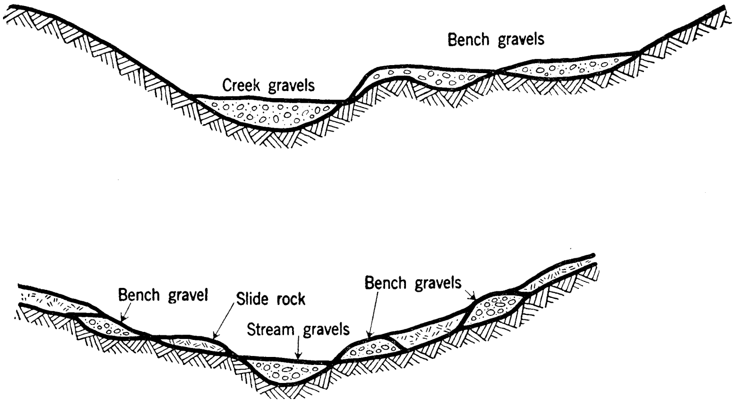

FIGURE 1.—Types of placer deposits (after Geological Survey).

In small-scale operations, prospecting and actual mining are often concurrent operations, the gold being recovered by panning, rocking, long toms, or small sluices. The gold pan is an indispensable part of the prospector's equipment.

If gold is found in and along the stream beds and their bars, it may be possible to trace it upstream and to old river benches above the present channel; prospecting of these benches should not be neglected.

Exploration of large placer deposits with a view to large-scale operation will be touched upon later.

LODE DEPOSITS

Prospecting for lode deposits begins with a search for mineralized outcrops, the key to the existence of which may often be found in mineralized float (pieces of rock from the lode that have been loosened by weathering and washed downhill).

Geologists now generally agree that most valuable mineral deposits have a magmatic origin and, therefore, that the most promising areas to prospect are in regions where intrusive igneous rocks are known to occur. The deposits themselves may or may not be in the igneous rocks; often they are in the invaded rocks at or near their contact with the intrusives. Another important fact to bear in mind is that ore deposits have been localized where channels existed for the entry of mineralizing solutions, and the rocks were of such nature or in such condition that they provided an environment favorable to the reception of these solutions and to the deposition of their contained minerals. Readily soluble rocks and rocks in which zones of fracturing have been developed are especially favorable. The latter condition is most likely to exist where the rocks have been contorted considerably by dynamic action, resulting in folding and faulting, and areas in which these phenomena occur are more favorable than others. Before embarking upon a prospecting campaign, it is important, then, to procure copies of geologic maps of the region, if such are available, and to study all reports on the known ore deposits so as to become familiar with the "habit" of ore occurrence in the region.

The details of prospecting procedure will depend on various conditions too numerous to be discussed in detail here. The work is simplified in regions where there are abundant exposures of rock and becomes more difficult where bedrock is covered with a thick soil, mantle rock, or heavy vegetation.

In prospecting, areas of exposed rocks are quite naturally examined first. Lodes frequently manifest themselves by their peculiar effects on topography. Where the lode material is more resistant to weathering and erosion than the surrounding country rock, the outcrop may stand up as an escarpment or ridge; and where the reverse condition exists, the outcrop may be expressed as a depression in the surrounding rocks.

In regions where weathering and oxidation take place at a faster rate than erosion, which frequently obtains in arid regions, the presence of mineralized lodes or bodies may be indicated by discoloration of the rocks. Since iron sulfides are commonly associated with primary deposits of ores of gold and the base metals, particularly copper, their oxidation products, red and yellow oxides of iron, often impart their color to the outcrop. Likewise, red oxides and green or blue carbonates of copper leave stains of characteristic color in the outcrop.

The character of outcrops has been discussed in more detail by Emmons.[13] For the purpose of this bulletin, it is sufficient to state on this point that peculiarities of topography or rock coloring, such as those mentioned, should be investigated when an area is prospected.

Since quartz and iron sulfides, or their oxidation derivatives, are, broadly speaking, the most common gangue minerals of ore deposits, quartz veins should be examined, especially if they contain oxidized iron minerals. In outcrops the quartz is likely to be honeycombed, owing to solution and removal of soluble minerals, particularly in metal-bearing lodes.

It oftens happens that outcrops of lodes are covered by soil and loose rock, in which case their discovery is more difficult. In such instances the first clue to the existence of a lode may be found in "float." The prospector will examine pieces of loose rock, especially those containing metallic minerals, and characteristic lode minerals, the most common of which is quartz.

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[13] Emmons, W. H., Outcrops of Ore Bodies: Min. and Sci. Press, vol. 99, 1909, pp. 751–754, 784–787.

Since the float must have moved downhill from its source in the lode, the prospector will work uphill from the point of his first discovery of float, searching for further similar indications. When he reaches a height where float no longer occurs, he may assume that he is over the lode or close to it.

In actual practice it is seldom as simple as this, and sometimes there may be an abundance of float that cannot be traced to its source in the lode. In some instances the float may have been carried long distances by glaciers or may be separated from its source by a rapidly cutting, intervening stream or by some other major change in topography that occurred since the float was loosened, thus obscuring its origin.

Gardner [14] has emphasized the importance of panning likely looking pieces of float in prospecting for lode gold. Since gold is a stable metal unaffected by ordinary solvents, it is more likely to be retained in the rock until freed by mechanical wearing away of other minerals than are most other metallic constituents. Thus, if the float be crushed, pulverized, and then panned, visible gold often will be found if the rock is gold-bearing. Although the pan is particularly applicable to gold, it may often be used to advantage to detect the presence of small amounts of other metallic minerals, most of which are heavier than the common gangue minerals, especially if they exist in the form of sulfides. As previously stated, the prospector must be familiar with the different metallic minerals, their visual appearance, and simple tests for their identification.

Having discovered the outcrop of a mineral-bearing lode either exposed on surface or revealed by trenching and test-pitting through overburden, it is generally advisable to confine initial work to tracing its extension laterally by a series of trenches, test pits, and shallow shafts. The latter should go deep enough to reach solid ledge actually in place.

All exposures of the vein or lode should be sampled systematically by methods discussed later in this bulletin, and a split of each sample should be sent to a reliable assayer. A map or maps should be prepared showing the property lines, position of trenches and pits, position of the vein or lode and its width at places where it has been exposed, and the position and length of each sample together with its assay value. Too often a record of this sort is not kept, to the later regret of the prospector when he endeavors to obtain capital for more comprehensive exploration and development of his property.

Before proceeding far with the work of proving his find, it is necessary, of course, to stake and record the claims and to comply with all provisions of the law governing location of mining claims in the country and State or Province in which the property is situated.

It is important that early work be confined as much as possible to the lode; far too often in the past a long crosscut tunnel or a shaft outside the lode has been driven or sunk to cut the projected lode at greater depth before enough shallow work has been done to determine its direction, dip, and general characteristics. If conditions are such that it seems necessary to prove the lode at depth before determining its persistence laterally, it is best in almost all instances to do this by means of a shaft or incline in the lode, following its irregularities, since all work done then gives additional essential information. The prospector ordinarily is not equipped financially to explore his find to the extent necessary to actually "block out" ore; it is doubly important, therefore, that every dollar spent bring a maximum return in the form of information regarding the deposit—its width, persistence along the strike and on the dip, and assay value.

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[14] Gardner, E. D., and Johnson, Fred W., Prospecting for Lode Gold, and Locating Claims on the Public Domain: Bureau of Mines Inf. Circ. 6843, revised, 1935, 18 pp.

REPORTING ON PROSPECTS

Usually it is necessary for the prospector to obtain financial assistance for further exploratory work before his property can be brought favorably to the attention of exploration and mining companies whose business it is to develop and operate mines. More often than not, such companies, the larger ones especially, are interested only in properties where ore is actually proved, and they will not go to the expense of an examination of a "raw" prospect.

This situation often leaves a gap between the preliminary work of the prospector who makes a discovery and does only the small amount of work upon it that can be done with his own hands or his own meager capital, and the work of the development company.

This gap frequently is filled by work paid for with funds supplied by local merchants and professional men or by a small syndicate, whose aim it is to block out enough ore to warrant forming an operating company or to interest the larger companies in exploitation of the property.

There are probably many instances in which examination of prospects of merit has been refused because of failure of the prospector to present essential facts.

Capital is risked in mining ventures to make a profit from operation. (This bulletin deals only with legitimate mine exploitation and not with promotional ventures, the main object of which is to make a profit from the sale of stock.) Hence, the first consideration is ore, if ore be defined as a body of mineral of such grade and occurring in such quantities that under the physical conditions affecting the cost of operation and in the locality in which it occurs it can be mined and sold at a profit.

Therefore, any report, to arouse the interest of mining capital, must present facts that indicate the existence of ore or the likelihood of its existence. Misrepresentation of facts by the prospector will not profit him, since such misrepresentation will only lead to condemnation of his property by an examining engineer who may be led to visit it; and this, in turn, might easily make it impossible to obtain a later examination. Furthermore, misrepresentation in a report can often be detected by an engineer because of the very means employed to cover the real facts.

If the prospector can afford to employ a reliable engineer to examine and report on his property, it is advisable to do so. Even though this engineer may be unable, because of the amount of work done, to make a favorable report, he may be able to indicate the best procedure for the prospector to follow in order to explore his ground further. In selecting the engineer, it is advantageous to employ one who is not only able and reliable but who also has the confidence of mining capital.

If unable to employ such an engineer, the prospector himself may be able to prepare a satisfactory report, if he will bear in mind the essentials. The importance of a map and the information that should be placed upon it have been mentioned already. To this should also be added any information the prospector can give concerning the mineralogy of the lode; the kind and condition of the wall rocks; relationship to intrusive rocks; structural features such as folding, shearing, and faulting; and surface weathering. There should also be given information on factors that would affect operating costs in the district, such as mode and cost of transportation to and from the property, climate, topography, vegetation, fuel and water supply, availability and type of labor, and other local conditons. Self-deception is a real hazard to a man preparing a report on his own property, and its possibility must be recognized and forestalled.

If ore is actually blocked out the prospector may include an estimate of its tonnage and grade; but if maps (in some instances cross sections also may be necessary) are included showing the essential facts as previously outlined, the examining engineer can and will make his own deductions regarding ore blocked out.

The essentials for a preliminary report have been discussed in more detail by Wright.[15]

EXPLORATION

Assuming that prospecting operations and preliminary exploration have revealed the existence of a lode or body of ore-grade material under geological conditions favorable for extensive deposition of ore and that money has been provided for a more thorough exploration program, the procedure to be followed will then depend upon a number of things, some of which already may have been partly determined by the preliminary work.

Thus, the type of deposit, its shape and dip, its relation to the surface topography, and the nature of the topography, and the character and condition of the lode and wall rocks are important considerations. The amount of money available for exploration, physical aspects of the property, accessibility for men and supplies, and fuel and water supply also may influence the procedure.

To direct exploration intelligently, careful records should be kept of results obtained, especially of all geological information, as the work progresses, in order that each step may be guided by the information obtained earlier. It is of great advantage to be able to obtain prompt assay returns on samples, and, except for small-scale operations within easy reach of a commercial assay office, it is advisable to have a laboratory and assay equipment on the job.

As in prospecting work, all ore exposures and mineralized material should be sampled and assayed systematically as the work progresses.

The object of an exploratory campaign may vary from one designed merely to prove enough ore to warrant starting a productive operation on a small scale to one aimed at determining approximately the total amount and grade of ore available as a basis for planning large-scale mining operations. In general, the former is applicable to relatively small high-grade deposits, whereas the latter applies to large deposits of low- or medium-grade material, including large low-grade placers.

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[15] Wright, Charles Will, Essentials for a Preliminary Report on a Small Lode-Gold Mine or Prospect With Notes on Sampling: Bureau of Mines Inf. Circ. 6748, 1933, 12 pp.

PLACER DEPOSITS

Although placer deposits are comparatively simple from a geological standpoint, accurate sampling and interpretation of sampling results are often difficult. Erroneous conclusions often have resulted from incomplete, inaccurate, or poorly interpreted sampling and have led to unprofitable operations.

Placer deposits are commonly explored by means of test pits or small shafts, by drifts, or by drilling. Drifts may be employed when the pay gravel is concentrated within a few feet of bedrock, is covered by thick overburden, or is exposed along the face of a bank or cut. Test pitting and drilling are the methods most generally employed, expecially where the deposit must be sampled from surface to bedrock or to the pay horizon, either because the method of mining proposed involves handling all of the gravel or because the gold occurs throughout the entire depth.

Where test pits can be employed they are satisfactory because of the large samples obtained and the opportunity to see the gravel as it occurs in the bank. Where a large volume of water is met in sinking this method becomes impracticable, not only because of the difficulty and cost of sinking but because the samples are salted by material that caves or washes in from the sides of the pit.

The drilling method employed is the drive-pipe method. Briefly, this method involves driving a pipe or casing into the gravel to a measured depth, removing the material inside the pipe (which constitutes a sectional sample of the material over that depth), driving again, and removing the material for another sample, and so on. Sometimes the material in the pipe may be loose enough so that it may be removed by simple bailing, but in other instances it may have to be broken up first by drilling inside the pipe with a churn drill.

However the samples may be obtained—by test pits, drifts, or drilling—the gold is washed from each sample by panning or rocking and is weighed. From the known or calculated volume of material in place, represented by the sample, and the weight of gold recovered from the sample, the weight and value per cubic yard may be estimated. Assays of the samples are not made (except, perhaps, occasionally as a check to determine the proportion of total gold in the sample recoverable by washing), and the gold actually recovered is used as a basis for estimating the value per cubic yard of the deposit.

It is not always a simple matter to determine accurately the amount of material in place represented by the sample or to be sure that salting or dilution has not taken place.

As in other types of deposits, large samples are more likely to be representative than small ones, and for this reason accuracy may often be improved by washing all the material excavated from test pits, shafts, or drifts rather than smaller samples cut from the sides. When such large samples are employed, a rocker ordinarily is used for washing, final clean-up being made with a pan.

Careful consideration should be given to the arrangement and spacing of test holes, which should depend on the characteristics of the deposit in each instance. In the case of channel deposits, long in one dimension and short in the other, it is customary to drill rows of holes across the channel normal to the long dimension, with the rows spaced at regular intervals. For blanket deposits, broad in both dimensions, the holes should be spaced at the corners of equal squares, checkerboard fashion.

The distance between rows and the holes in the rows in channel deposits and likewise the size of the squares in blanket deposits will depend on the continuity of the pay gravel and its degree of uniformity in grade. Several channel deposits have been drilled with holes 150 feet apart in rows 1,000 to 1,500 feet apart.[16] This distance between rows would seem to be too great for erratic deposits.

Engineers of the Reconstruction Finance Corporation, which has granted loans for dredging operations in the western United States, like to have drill holes in narrow deposits not more than 100 feet apart in rows not more than 1,000 feet apart and in bench or blanket deposits on coordinates not less than 400 feet apart.

The best engineering practice probably would be to space the first holes or shafts at about these maximums, with later check-holes between, spaced as indicated to be necessary by the distribution of the gold and the shape of the deposit as shown by the earlier holes.

Tests should be carried down to and into bedrock a short distance, since coarse gold is often concentrated in crevices that commonly occur in the top of the bedrock.

The formula for calculating the value per cubic yard represented by each drill sample is as follows:

$$\frac{C\times M}{A\times D}\times 27=\text{theoretical value per cubic yard}$$

where,

27 = cubic feet in a cubic yard,
C = value of gold in cents per milligram,[17]
A = area of drive shoe, square feet,
D = length of sample in feet,
M = milligrams of gold recovered from sample.

The average value for each hole is obtained by averaging the values of the individual samples, weighted when not representing equal yardages; and the average for the entire deposit likewise is determined according to mathematical principles by averaging the weighted values of all the holes. The yardage is determined from the depths of the holes and the areas that each represents. In combining the values and estimating the yardages, experience and good judgment are required. The estimator must consider the type of the deposit, the distribution of the gold, and the method of mining to be employed and its limitations with respect to the degree to which sample recoveries can be duplicated in actual mining practice. He will have to make allowances dictated by his judgment and experience.

Table 1 gives a comparison of estimates based on drive-pipe samples with actual dredge recovery.

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[16] Gardner, Walter H., Prospecting for Placer Gold with the Keystone Drill; Drilling for Placer Gold: Keystone Driller Co., Beaver Falls, Pa., undated, pp. 9-87.

[17] This value will depend on the fineness of the gold as determined by assay; for gold 850 fine, this is 0.06 cent with the price of gold $20.67 per ounce, or about 0.097 cent with gold at $35 per ounce.

TABLE 1.—*Comparison of estimates based on drive-pipe samples and actual dredge recovery* [1]

1	2	3	4	5				
Example	Area, acres,	Average depth, feet	Acreage per hole	Size of shoe, inches,	Area factor used	Value, per cubic yard, cents		Recovery, percent
						Estimate from samples	Dredge recovery	
Oregon [2]	121.0	18.0	2.4	7½	0.3333	16.8	15.63	93.0
California	118.5	(?)	3.1	7½	.3068(?)	29.88	31.55	105.6
California A	173.5	22.5	3.2	7½	.27	6.8	7.82	115.0
California B	84.0	44.5	4.2	7½	.27	5.9	6.70	113.0
California C1	183.0	51.8	1.5	7½	.30	11.1	9.64	87.0
California C2	106.0	60.6	2.6	7½	.30	11.2	9.44	84.0
California C3	135.0	56.4	2.3	7½	.30	11.6	11.30	97.0
Average						9.48	9.12	96.2
California:								
1918	19.94	32.1	1.8			10.39	10.64	102.4
1919	20.90	34.4	2.1			9.69	9.22	95.2
1920	20.43	29.8	2.9			10.69	14.34	134.0
Natomas	421.0			7½	.30	10.25	11.39	111.1
Montana	300.0	40.0	3.9			15.83	13.55	85.6
California	559.0	22.5	1.0	7½	.27	7.58	9.61	126.7
Do	420.5	35.9	2.9	7½	.27	7.25	8.18	112.9

[1] Data taken from Drilling Results and Dredging Returns, by Charles W. Gardner: Eng. and Min. Jour., vol. 112, No. 17, Oct. 22, 1921, pp. 646–649, et seq.

[2] Area divided into 8 blocks. Values estimated for individual blocks gave much wider variations from actual dredge recovery than the average, ranging from 49.4 percent in excess of the estimate to 68.2 percent less than the estimate. Method of calculation: Cross sections on lines of holes.

The exploring and sampling of placer deposits have been discussed in a number of technical articles, to several of which the reader is referred.[18]

The cost of sampling placer deposits and even the cost per foot of test pitting and drilling varies within wide limits. The cost per cubic yard of gravel tested will depend on the thoroughness of the job, the number of cubic yards tested per foot of test pit or drill hole, the cost per foot of test pitting and drilling, and the depth of the gravel.

The cost per foot of test pitting or drilling will depend on the location, the characteristics of the gravel, the size and distribution of boulders, presence or absence of water, degree of consolidation or cementation, and other physical conditions.

Table 2 presents a few typical sampling costs.

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[18] Janin, Charles, Gold Dredging in the United States: Bureau of Mines Bull. 127, 1918, 226 pp.

Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bureau of Mines Bull. 259, 1927, 236 pp.

Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bul. 356, 1932, pp. 6–12.

Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. Pt. I. General Information, Hand-Shoveling, and Ground-Sluicing: Bureau of Mines Inf. Circ. 6786, 1934, pp. 23–46.

Vanderburg, W. O., Placer Mining in Nevada: Nevada State Bureau of Mines and Mackay School of Mines, Bull. 4, vol. 30, May 15, 1936, pp. 29–34.

Gardner, Walter H., Prospecting for Placer Gold with the Keystone Drill: Drilling for Placer Gold: Keystone Driller Co., Beaver Falls, Pa., undated, pp. 11–40.

Stines, Norman C., The Prospecting and Valuing of Dredging Ground: Min. and Sci. Press, Feb. 10, 1906, p. 90.

Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Bureau of Mines Inf. Circ. 6612, 1932, pp. 5–6.

Patmon, Charles G., Methods and Costs of Dredging Auriferous Gravels at Lancha Plana, Amador County, Calif.: Bureau of Mines Inf. Circ. 6659, 1932, p. 3.

Requa, Lawrence K., Description of the Property and Operations at the Lewiston Dredge, Lewiston, Calif.: Bureau of Mines Inf. Circ. 6660, 1932, pp. 3–5.

Ross, Charles L., and Gardner, E. D., Placer-Mining Methods of E. T. Fisher Co., Atlantic City, Wyo.: Bureau of Mines Inf. Circ. 6846, 1935, pp. 2–3.

Sawyer, Dwight L., Sampling a Gold Placer: Eng. and Min. Jour., July 1932, pp. 381–383.

Avery, William W., Computing Drill-Hole Data in Placer Prospecting: Eng. and Min. Jour., vol. 129, No. 10, May 1930, pp. 493–495.

TABLE 2.—*Typical placer-sampling costs*

Location and year	Sampling method	Character of deposit	Average depth, feet	Cost per foot
Folsom, Calif., 1931[1]	Gasoline-driven drill	Gravel with much clay	68.0	[2] $2.98
Waldo, Oreg., 1931[1]	do	do	86.7	[2] 3.665
Sumpter, Oreg., 1908–10[1]	Steam-driven drill	Loose gravel, 16 percent soil.	19.5	[2] 2.31
Salmon, Idaho, 1906[1]	Shafts, untimbered	Shallow gravel	13.7	[3] 3.46
Snelling, Calif., 1931–32[1]	Shafts, caissons	Loose gravel, 20 percent soil, much water.	16.9	[4] 9.85
Skull Valley, Ariz., 1932[1]	Shafts, 2 by 3 to 4 feet		3.5	(5)
Montana[1]	Shafts, 4 by 4 feet and 4½ feet circular.	Moderately fine gravel, stood well.	35.0	[6] 1.00
Do[1]	Gasoline-driven drill	3 feet of gravel covered with black mud.	50.0	[7] .50
Colorado, 1930–31[1]	Drill	Bouldery	50–60	2.87
Oroville, Calif., 1932[1]	do		50	2.50
Dawson, Yukon, 1907	Shafts, no timber	71 percent frozen muck, 29 percent frozen gravel.	24.3	[2] 3.50
Amador County, Calif., 1928–29.[8]	Gasoline-driven drill	40 percent sand, 40 percent gravel, 20 percent loam and clay.	21.8	[9] 1.00
Lewiston, Calif., 1928[10]	Steam-driven drill	Easy digging, few large boulders, some clay.	35–40	[2] 7.23
Arizona[11]	Shafts, 3½ by 5½ feet, cribbed.	Many large boulders	12.0	[12] 3.00–6.00
Angels Camp, Calif.[13]	Gasoline-driven drill	40 to 50 feet rhyolite capping; cobble and compressed gravel.	134.0	[7] 6.11
Nome, Alaska[14]	Shafts and pits untimbered, 3½ by 3½ feet.	Frozen gravel and muck	30–50	[7] 3.50
Do.[14]		do	40–50	2.00
Do.[14]	Shafts timbered, 4 by 5 feet.	Partly frozen gravel and muck.	90	8.00
Ophir Creek, Alaska[14]	Pits 3 by 6, untimbered	Not frozen	5–14	1.75
Kongaroh River, Alaska[14]	Pits, untimbered	Frozen gravel and muck	5–6	.65
Otter Creek, Alaska[14]	Pits, 3½ feet diameter, untimbered.	Frozen gravel	14–17	4.00
Willow Creek, Alaska[14]	Pits, 3½ feet diameter, untimbered.	Frozen muck and gravel	16–18	3.00
Ruby district, Alaska[14]	Shafts, 4 by 6 feet, untimbered.	do	80	6.00
Fairbanks district[14]	Shafts, 5 by 7 feet, top, 5 feet, timbered.	Partly frozen muck and gravel.	20–22	2.50
Do.[14]	Shafts, 4 by 7 feet, top, 20 feet, timbered.	Frozen gravel and muck	100	8.00
California, 1931–32	Rotary drill, 2½ feet diameter.	Dry gravel	25	.35

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[1] Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. Part 1. General Information, Hand-Shoveling, and Ground-Sluicing: Bureau of Mines Inf. Circ. 6786, 1934, pp. 26–39.

[2] Including sinking and panning.

[3] Sinking $2.01, sampling $1.45.

[4] Sinking $7.48, sampling $2.37.

[5] $0.65 per cubic yard excavated; Mexican labor, $3.50 per day.

[6] Sinking only, on contract, long hours; $3.50 per day earned.

[7] Drilling only.

[8] Patmon, Charles G., Methods and Costs of Dredging Auriferous Gravels at Lancha Plana, Amador County, Calif.: Bureau of Mines Inf. Circ. 6659, 1932, p. 3.

[9] Approximate cost of drilling only.

[10] Requa, Lawrence K., Description of the Property and Operations at the Lewiston Dredge, Lewiston, Calif.: Bureau of Mines Inf. Circ. 6660, 1932, pp. 3–6.

[11] Sawyer, Dwight L., Sampling a Gold Placer: Eng. and Min. Jour., July 1932, pp. 381–383.

[12] Labor only.

[13] Steffa, Don, Gold Mining and Milling Methods and Costs at the Vallecito Western Drift Mine, Angels Camp, Calif.: Bureau of Mines Inf. Circ. 6612, 1932, pp. 5–6.

[14] Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bureau of Mines Bull. 259, 1927, p. 35.

LODE DEPOSITS

The primary object of exploring mineral deposits in the sense that the term "exploration" is used in this bulletin is to prove up ore following the discovery of a lode or of material of ore grade. Hence, exploration is concerned primarily with ascertaining tonnages and grades of ore, which involves locating ore shoots and determination of their size by exposing the ore in mine workings or in drill holes, and procuring for analysis samples from the exposures thus made.

These data are employed by the engineer or geologist in estimating reserves.

The normal sequence of exploration, mine development, and mining or ore extraction is seldom, if ever, a series of disconnected operations, but rather of overlapping ones—overlapping in the sense that exploration usually continues with varying intensity during development and mining, and some development, at least, continues until the ore deposit is nearly worked out.

Thus, in operating mines, exploration for extensions of known ore bodies, both laterally and at greater depth, and the search for "blind" lodes in the walls of those being worked is continued with a view to finding new ore bodies to supplement those being extracted and so prolong the profitable life of the enterprise. In such work, as well as in exploration of virgin areas, experience, a high type of trained intelligence, and knowledge of the habits of ore deposits are necessary for the direction of the work if the best results are to be obtained at minimum cost.

The best method of exploration in any given instance will depend on a number of factors, such as position of the lode with respect to surface topography; its shape, size, and dip; the relation of ore shoots to structural forms or to certain types of rocks; and other geological features. Many of these may not be known at the outset. The picture gradually unfolds as work progresses, revealing basic relations that guide further exploration.

Exploration may be conducted from the surface, from underground workings, or both, depending on local conditions. In unexplored areas where ore outcrops have been found, preliminary exploratory work may be confined to surface trenching, test-pitting, or, where the topography is favorable, to adits driven on the lode.

Often, however, valuable ore deposits do not outcrop, and there may be no surface indication of ore below, aside from structural and other geological conditions under which ore deposits have been found previously in the same general region. Under such conditions, diamond or churn drilling often has been employed to probe for possible blind ore bodies. Thus, in the Kansas-Oklahoma section of the Tri-State lead and zinc district, there were no surface indications of ore whatever where most of the largest productive mines were developed. They were found by widening the field of exploration around earlier discoveries made in the Southwestern Missouri district where the ore had been exposed by erosion of the overlying rocks. Subsequently, the relationship of the ore bodies to zones of shearing, shattering, and brecciation was worked out and served as a guide to further exploration in the area.[19]

In active mining districts, important discoveries have been made from time to time adjacent to productive mines in areas long considered unpromising. Thus, the rich silver ores of the Silver Belt, formerly known as the Dry Belt, in the Coeur d'Alene district were developed in an area close to large productive mines, which was long considered unfavorable; the rejuvenation of the Idaho-Maryland mine, abandoned for years, was brought about by finding important vein extensions close to the old workings; the southeastern part of Tintic district in Utah, now the most productive part of the district, was long considered unpromising; many other examples might be cited. In large ore-producing districts favorable areas for exploration are often along extensions of the controlling formations or geological structures, similar parallel zones, or where the geological conditions are similar to those of the known deposits. The same may be said of more restricted areas in individual mines.

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[19] Fowler, George M., and Lyden, Joseph P., The Ore Deposits of the Tri-State District (Missouri-Kansas-Oklahoma): Trans. Am. Inst. Min. and Met. Eng., vol. 102, 1932, pp. 206–251.

Where the geology is especially complex, it is sometimes impossible to class any ground as barren until so proved by actual exploration. In other districts, however, long experience and much exploratory work have established the fact that the ores are restricted to certain kinds of rocks or to certain definite and well-defined structures. It is therefore obvious that a knowledge of geology and of conditions influencing ore deposition is invaluable in conducting an exploratory campaign.

Exploration may be by excavation (trenches, pits, adits, shafts, drifts, crosscuts, and raises), by drilling, or by both. In recent years geophysical methods have been employed to furnish additional criteria upon which to base interpretation of other geological evidence. As progress is made in the science of geophysics, it doubtless will be used more and more in exploratory work.

Summaries of exploration methods and costs in many different districts and mines have been presented in earlier Bureau of Mines bulletins.[20]

Details of practices at individual mines are given in the various information circulars comprising the series on mining methods and costs, many of which are cited in the above bulletins.

EXPLORATORY WORKINGS

It has been stated already that in prospecting, work should be confined as much as possible to the vein or ore body with a view to obtaining the maximum amount of information about it at a minimum cost. This applies also to exploratory workings, especially in the earlier stages before the existence of enough ore to justify mining operations or the continuity of the ore bodies has been demonstrated. Too often long crosscuts have been started, designed to tap anticipated downward extensions of a vein exposed at surface, only to find that the ore was not where expected, whether because of petering out, faulting, or change in direction or dip.

If considerable is already known about the boundaries of the ore, exploratory workings may be planned with a view to their utility and adequacy for future use during the productive period of the enterprise. Sometimes, however, it is possible to give too much weight to future usefulness of exploratory openings, particularly if it has not yet been demonstrated that deposits of economic importance are present. It may be advisable, then, to ignore the future usefulness of these openings and locate them to obtain maximum information at minimum cost, even though it may mean that for efficient mine operation new shafts, adits, and other main openings will have to be excavated later on at considerable cost. Thus, an inclined shaft may serve to develop a deposit dipping at a low angle much more quickly and at much less cost than a vertical shaft, although a vertical shaft might be better adapted to operating purposes later when the mine enters production.

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[20] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, 155 pp.

Jackson, Chas. F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, pp. 38–50.

Jackson, Chas. F., Knaebel, John B., and Wright, C. A., Lead and Zinc Mining and Milling in the United States; Current Practices and Costs: Bureau of Mines Bull. 381, 1935, pp. 47–79.

Gardner, E. D., Johnson, C. H., and Butler, B. S., Copper Mining in North America: Bureau of Mines Bull. 405, 1938, 300 pp.

Therefore, an engineer who employs an inclined shaft in the vein or other main entrance for exploratory purposes, poorly planned from an operating viewpoint, should not necessarily be criticized if a mine is proved up and these openings have to be replaced later by others better suited to an operating property. However, in a producing mine or partly developed property, exploration work should be done with a view to utilizing it later for development and production to save duplication of expenditures.

The cost per foot of exploratory work at an undeveloped property is generally at least twice as much as at producing mines, for several reasons. In the first place, supervision, shop, camp maintenance and operation, other general charges, pumping, hoisting, power, etc., are all chargeable to the exploratory workings instead of being prorated between production, exploration, and development, as at an operating mine. Furthermore, exploration work often is conducted in localities where transportation and power costs are high and where living quarters must be provided and maintained, all of which add materially to unit costs.

Table 3 presents data on costs of transporting men, equipment, supplies, and ore in a number of mining districts, and figure 2 shows costs of trucking ore plotted against length of haul.[21] The transportation of equipment and supplies constitutes a somewhat different problem from that of transporting ore, but the latter is indicative of the variations in costs in different localities and under different conditions and should approximate the costs of the former. Table 4, taken from Gardner,[22] gives some costs of packing ore in the Western States and Canada.

Table 5 presents some typical costs of exploratory workings (shafts, drifts, crosscuts, and raises).

EXPLORATION BY DRILLING

Drilling is employed extensively, either as the principal exploratory method or to supplement exploration by underground and surface workings. Three principal types of drills are employed for this purpose—churn drills, core drills, and hammer drills. In ground too soft to core and in which a churn drill would stick, a rotary drill with a fish-tail bit is useful for obtaining sludge samples.

In churn drilling, a string of tools with a cutting bit at the lower end is suspended from a rope or cable and is alternately raised and dropped by hand or power-driven mechanism, chopping a hole in the rock. Water is used in the hole during drilling, and the cuttings are removed with a bailer, usually at regular intervals of about 5 or 10 feet of hole. These cuttings constitute samples of the material passed through and may be examined mineralogically and assayed.

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[21] Gardner, E. D., Costs of Trucking and Packing Ore in Western Gold-Mining Districts: Bureau of Mines Inf. Circ. 6898, 1936, 17 pp.

[22] See footnote 21.

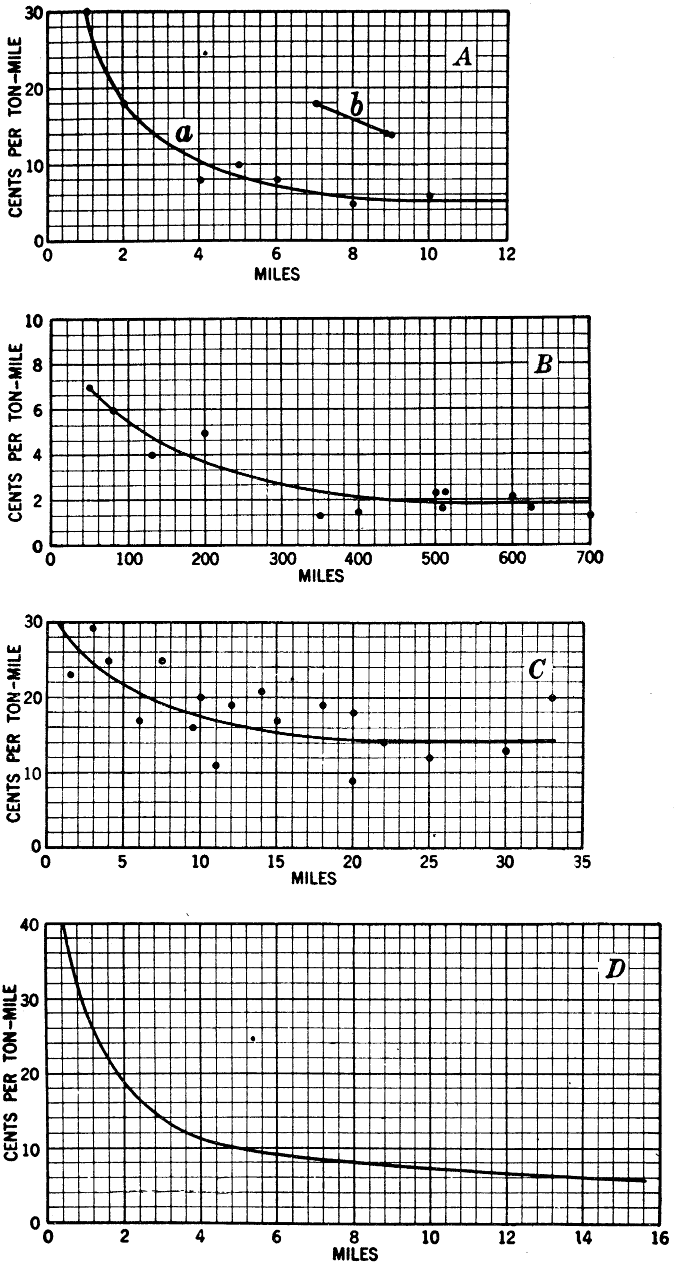

FIGURE 2.—Curves showing ore-trucking costs. *A*, Oatman and Katherine districts, Arizona, dirt roads. *B*, Long-distance trucking in small lots, California and Arizona. *C*, Nevada, fair dirt roads, irregular lots of ore. *D*, Arizona and California, good dirt roads, steady hauling.

TABLE 3.—*Costs of transportation of men, equipment, supplies, and ore*

District	Year	Mode of transportation	Length of one-way trip, miles	Cargo		Cost per ton	Cost per ton-mile
				Total	Per trip		
Kenora, northern Ontario [1]	1930	Tractor and sleighs	120	400 tons supplies and heavy equipment.	80 tons	$84.91	[2] $0.708
Do.[1]	1930	Airplane	100		1,500 pounds	[3] 0.045	.900
Do.[1]	1930	do	100	Passenger		[4] 85	
Matachewan, Ontario [5]	1930–31	Tractor, teams, and sleighs	42	700 tons		[6] 45	[2] 1.07
Do.[5]	1931–32	Tractor and sleighs	42	1,412 tons		[6] 27	[2] .64
Do.[5]	1931–32	Airplane	30+	45 tons	1,150 pounds, maximum.	[3] .06	[2] 3.75
Do.[5]	1931–32	do	30+	Passenger		[7] 10	
Patricia, northern Ontario [8]	1928	Tractor and sleighs	120	Heavy supplies		140	1.16
Do.[8]	1928	Airplane	100+	Perishable foodstuffs		[3] 0.12	
Ontario [8]	1930	Teams	14 (rough corduroy road for 11 miles).	Supplies		25	
Do.[8]	1932	Water route, one short portage	27	do		7–8	0.26–0.30
Quebec	1935	Tractors and sleighs	25	Supplies in and concentrates out	12 tons	6.50	[9] .26
Do	1935	Water route (barges)	75	do	30 tons	3.00	.04
Jarbidge, Nev.[10]	1931	Motortruck	70 (dirt roadway)	Supplies		[6] 20.00	.28
Black Canyon, Ariz.[11]	1931	do	15	Concentrates and supplies	3 to 4 tons	0.75	.05
Farrel, Nev.[12]	1935	do	35	do		4.00	.11
Rosebud, Nev.[12]	1935	do	9	Ore		[6] 2.00	.22
Do.[12]	1935	do	30	do		4.00	.13
Sacramento, Nev.[12]	1935	do	6	do		[6] 2.00	.33
Seven Troughs, Nev.[12]	1935	do	26	do		[6] 3.85	.15
Eureka, Nev	1934	do	4	do		[6] 1.00	.25
Tonopah, Nev	1934	do	25	do		[6] 2.50	.10
Manhattan, Nev	1934	do	50	do		5.50	.11
Copper Basin, Nev	1934	do	10½	do		2.00	.19
Do	1934	do	9½	do		1.50	.16
Argenta, Mont	1936	do	18	Ore and supplies		[6] 2.00	.11
Bleuming, Mont	1936	do	12½	Ore		[6] 1.75	.14
Black Canyon, Ariz.[13]	1936	do	14½	do		[6] 2.00	.138
Peck, Ariz.[13]	1936	do	23	do		4.00	.174

[1] Keast, A. J., and Jackson, Chas. F., Method and Cost of Exploring, Equipping for Development, and Developing the Central Patricia Group of Claims, Northern Ontario: Bureau of Mines Inf. Circ. 6681, 1933, 13 pp.

[2] Per ton-mile in-bound; no credit for small return loads.

[3] Per pound.

[4] Per round trip. A few years later, rates were lower.

[5] Emens, W. H., and Jackson, Chas. F., Methods and Costs of Developing and Equipping the Ashley Gold Mine, Matachewan Gold District, Ontario: Bureau of Mines Inf. Circ. 6707, 1933, 28 pp.

[6] Contract price.

[7] Each one way.

[8] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[9] Load out and load in each trip.

[10] Park, John Furness, Mining Practices, Methods, and Costs at Elkoro Mines, Jarbidge, Nev.: Bureau of Mines Inf. Circ. 6543, 1931, 12 pp.

[11] Minton, David C., Jr., Cost of Equipping and Developing a Small Gold Mine in the Bradshaw Mountains Quadrangle, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6735, 1933, 10 pp.

[12] Vanderburg, William O., Reconnaissance of Mining Districts in Pershing County, Nev.: Bureau of Mines Inf. Circ. 6902, 1936, 57 pp.

[13] Guiteras, Jos. R., Gold Mining and Milling in the Black Canyon Area, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6905, 1936, 51 pp.

TABLE 4.—*Costs of packing ore in Western States*

Place	Distance one way, miles	Trips daily	Tons daily	Kind of animals	Number of animals per ton	Condition of trail	Drop in elevation, feet	Year	Cost per ton	Cost per ton-mile	Remarks
Dos Cabezas, Ariz	2	3		Burros	7	Fair	1,000	1934	$1.50	$0.75	Contract.
Do	1	5	5	do	7	do	700	1935	1.00	1.00	Do.
Ruby, Ariz	5	1		do	8	do		1935	3.50	.70	Do.
Wickenburg, Ariz	1	5	4	do	7	Poor	1,000	1935	3.00	3.00	Company account; 10 burros; 2 men on horses.
Superior, Ariz	½ to 1½		25	do		Fair	800	1934	1.00 to 1.50	1.00 to 2.00	Contract.
Prescott, Ariz	¾	5		do		Steep		1934	1.25	1.66	Do.
Do	1¼	3	3	do	8	do	1,000	1934	2.00	1.60	Do.
Camp Bird, Colo	3	2	[1] 10	Mules	5 to 6	Good	1,500	1934	[2] 6.00		28 to 30 mules, 2 horses, and 2 men. Contract.
Questa, N. Mex	¼			Burros		Fair	300	1930	1.00 to 1.50	4.00 to 6.00	Contract.

[1] Concentrate.

[2] Includes a 6-mile wagon haul; up freight, $8 per ton.

TABLE 5.—*Unit costs of exploratory workings*

Name and situation of property	Year	Before or during productive stage	Kind of work	Costs per foot of exploration work						
				Labor	Explosives	Power	Timber	Other supplies	General charges	Total
Macassa, Ontario, Canada [1]	1931–32	Before	9- by 17-foot shaft sinking 2,489 feet.	$36.175	$6.367	$6.581	$6.087	$13.208	$2.512	[2] $70.930
Central Patricia, Ontario, Canada.[3]	1930	do	6.5- by 16-foot shaft, 527 feet	[4] 23.558	7.546	[5] 13.579	[5] 9.471	[5] 24.210	13.026	[6] 78.364
Do	do	do	4-shaft station, 89.3 feet							[6] 68.13
Do	do	do	1,776 feet of drifts	[4] 6.210	3.498	[5] 1.765		[5] 6.572	[5] 3.181	[6] 21.226
Do	do	do	1,019 feet of crosscuts	[4] 6.567	3.543	[5] 2.025	[5] .018	[5] 7.672	4.263	[6] 24.088
Ashley, Ontario, Canada [7]	1930–32	do	6- by 17-foot inclined shaft, 436 feet	Shaft 44.19 Other 25.74	5.46	[8] 9.55	13.15	15.23	5.77	[9] 119.09
Do	do	do	Drifts and crosscuts, 4,308 feet	Direct 6.24 Other 5.08	2.91	2.09		5.13	1.09	[9] 22.54
Do	do	do	Raises, 638 feet	Direct 6.79 Other 5.85	3.68	2.42		4.62	1.22	[9] 24.58
Mine No. 5 [10]	1928	do	6- by 16-foot shaft, 556 feet	[11] 30.263	5.380	4.896	5.057	[5] 12.261	10.639	[9] 68.496
Do	do	do	Drifts and crosscuts, 2,579 feet	[4] 5.445	3.417	1.436		[5] 5.856	3.248	[9] 19.402
Mine No. 6 [10]	do	do	6 feet 8 inches by 14 feet 9 inches shaft, 475 feet.	[11] 33.507	5.164	3.010	6.947	[5] 8.929	8.143	[9] 65.700
Do	do	do	Drifts and crosscuts, 6,956 feet	5.776	2.486	1.725		[5] 3.969	2.954	[9] 16.910
Reno, British Columbia [12]	Prior to 1928	do	Drifts, 1,375 feet; raises, 165 feet							19.42
Do.[12]	1931–32	During	Tunnel, 1,086 feet	[13] 5.81	(13)	1.63	[14] .66	.92	3.80	12.82
Golden Belt, Bradshaw Mts., Ariz.[15]	1931	Before	Inclined shaft on 14° slope, 150 feet.	3.29	1.85			1.08		6.22
Gold Coin, Mont.[16]		do	4- by 5-foot shaft, 140 feet							16.00
Rabbit, Oreg.[16]		do	4½- by 7-foot shaft, 75 to 140 feet							40.00
Bullion Dome, Oreg.[16]		do	4- by 6-foot 65° shaft, 32 feet							7.70
Gold Hill, Utah [16]		do	6- by 8-foot shaft, 150 to 316 feet							[17] 15.50
Gold Queen, Utah [16]		do	3½- by 7-foot shaft							7.01
Mossback Extension, Ariz.[16]		do	7- by 10-foot shaft, 65 to 104 feet							[18] 14.35
Maxwell No. 1, Ariz.[16]		do	4- by 6-foot shaft, 40 feet	15.00					1.00	16.00
Maxwell No. 2, Ariz.[16]		do	5- by 9-foot shaft, 16 feet	20.00					1.00	21.00
Comstock Dexter, Ariz.[16]		do	6½- by 12-foot shaft, 95 feet							[19] 32.03

[1] Howes, G. A., and Jackson, Chas. F., Shaft-Sinking Methods and Costs of Plant and Equipment at Macassa Mine, Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6674, 1932, 10 pp.

[2] Does not include $66,228 cost of clearing, buildings, plant, and equipment, much of which was of service after the mine reached the production stage.

[3] Keast, A. J., and Jackson, Chas. F., Method and Cost of Exploring, Equipping for Development, and Developing the Central Patricia Group of Claims, Northern Ontario: Bureau of Mines Inf. Circ. 6681, 1933, 13 pp.

[4] Direct underground labor only.

[5] Materials, supplies, and indirect labor.

[6] Does not include $33,964 for transportation of supplies and equipment, $11,371 for buildings, $60,847 for surface plant equipment, and $11,077 for underground equipment, much of which was applicable to lateral exploration and later mine operation.

[7] Emens, W. H., and Jackson, Chas. F., Methods and Cost of Developing and Equipping the Ashley Gold Mine, Matachewan District, Ontario: Bureau of Mines Inf. Circ. 6707, 1933, 28 pp.

[8] Cost of boiler fuel only.

[9] Cost of transportation of equipment and supplies; camp and plant buildings and equipment not included.

[10] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[11] Shaft labor, timber-framing, pipe-fitting, and decking only.

[12] Mateer, T. J., What It Cost to Open the Reno Mine: Eng. and Min. Jour., vol. 135, March 1934, pp. 111-115.

[13] Breaking ground (labor and explosives) and workmen's compensation.

[14] Timbering (labor and timber).

[15] Minton, David C., Jr., Cost of Equipping and Developing Small Gold Mines in the Bradshaw Mountains, Quadrangles, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6735, 1933, 10 pp.

[16] Gardner, E. D., and Johnson, C. H., Mining and Milling Practices at Small Gold Mines: Bureau of Mines Inf. Circ. 6800, 1934, 31 pp.

[17] Direct cost only; no overhead charged.

[18] All costs included, but labor was only $1.50 per day and timber was salvaged material.

[19] Including half the pay of a blacksmith but excluding supervision and overhead.

Core drills are rotary drills that cut an annular groove about a central core of rock. This core is broken off and removed in sections as the hole is deepened, each section of core comprising a sample that can be analyzed and likewise reveals the composition, texture, and structure of the rocks penetrated. The cores may be split in half longitudinally by means of a core splitter, one half being used for assay purposes and the other retained as a more or less permanent visual record of the formations for inspection and correlation with the cores from other holes.

There are three general types of core drills: (1) Those employing a bit of soft steel set with a number of cutting elements, usually carbons ("black diamonds") or bort, (2) those of the shot-drill type, in which the annular groove is cut by grinding the rock with chilled shot or other hard material under a rotating shoe or bit, and (3) those employing a toothed cutter (sometimes used in soft rocks).

The hammer drill is employed principally in underground work. It is the common heavy rock drill using ordinary hollow drill steel and standard bits, except that for drilling long holes the machine usually is equipped with strong independent rotation and sectional steels are employed. The sludge is caught as it issues from the hole during drilling and is used as a sample or series of samples for visual examination and assay.

Each type of drill has its limitations and its particular fields of usefulness. Although short holes occasionally are drilled at steep angles with churn drills, they are not suited to drilling other than vertical holes. One great advantage of the diamond drill is that holes may be drilled at any angle from the vertical, up or down. Deep diamond-drill holes usually deviate from a straight line, and methods have been devised to straighten curved holes. It is also possible to turn a hole at any desired depth by means of deflecting wedges, and there are interesting examples of drilling several branching holes from a predetermined point in an original hole.

The churn drill has found special favor in drilling a number of the large so-called disseminated "porphyry" copper deposits in southwestern United States. In the Tri-State lead and zinc district the churn drill has been used almost exclusively. The diamond drill is not well-adapted to the work in this district because, on account of the cherty, fractured, often bouldery and vuggy nature of the ores and surrounding wall rocks, the percentage of core recovered is low and consumption of carbon or bort by excessive wear, breakage, and loss is high. In the Lake Superior iron-ore districts a combination diamond- and churn-drill machine, known as the Mesabi rig, has been used extensively. When drilling is done in soft formations that will break and thus not return a core, vertical holes may be drilled with a chopping bit screwed on the end of the regular string of rods. When passing through hard formations, the chopping bit is removed and replaced by a diamond bit.

In the Southeastern Missouri lead belt the diamond drill has been employed almost exclusively for drilling the disseminated ores, which are homogeneous, give a good core recovery, and drill easily with this type of drill. In the Mascot (Tenn.) area both churn and diamond drills have been used extensively.

Until quite recently, diamond-drill bits were set almost exclusively with carbons ("black diamonds"), but today bort is used more than carbons in the United States and Canada. About 4 to 10 or more times as many stones are required when bort is used as when carbon is used, and the cost of setting the bits is therefore greater. The cost per carat for bort, however, is about one-tenth that of carbon, or less. In many places it has been found that a bit set with bort will drill more footage than one set with carbon, that it will give better core recovery in some kinds of ground, and that the cost per foot of hole is much less. The loss of a few stones by breakage or of an entire bit is not nearly as costly as when carbon is employed.

Standard diamond-drilling outfits are built with steam, gasoline, electric, or compressed-air drive. Ordinarily, they are built in sizes ranging from those taking a ⅝-inch core and limited to 150 feet of drilling depth to machines that will take a 2⅛-inch core and drill to a depth of 5,000 feet. Other core sizes and capacities also are built for special work. The smaller outfits are sectionalized (knock-down), so that the largest and heaviest individual part can be transported under the most difficult conditions. The smaller rigs can readily be taken into and operated in stopes and drifts.

In 1930 the Diamond Core Drill Manufacturers Association (United States and Canada) established new standard sizes of bits and casing for four sizes of bits based upon a series of four nesting casings, so that the bit for one size hole will pass through the casing that will go in the next larger size hole. These sizes are as follows: [23]

Nominal dimensions

Size designation		Casing, outside diameter	Casing coupling		Casing bit, outside diameter	Core-barrel bit, outside diameter	Drill rod, outside diameter	Approximate diameter of hole made by core-barrel bit [1]	Approximate diameter of core
Casing, casing coupling, casing bits, core-barrel bits	Rod, rod couplings		Outside diameter	Inside diameter					
		Inches	*Inches*	*Inches*	*Inches*	*Inches*	*Inches*	*Inches*	*Inches*
EX	E	$1\frac{13}{16}$	$1\frac{13}{16}$	$1\frac{1}{2}$	$1\frac{27}{32}$	$1\frac{7}{16}$	$1\frac{5}{16}$	$1\frac{1}{2}$	$\frac{7}{8}$
AX	A	$2\frac{1}{4}$	$2\frac{1}{4}$	$1\frac{29}{32}$	$2\frac{5}{16}$	$1\frac{27}{32}$	$1\frac{5}{8}$	$1\frac{7}{8}$	$1\frac{1}{8}$
BX	B	$2\frac{7}{8}$	$2\frac{7}{8}$	$2\frac{3}{8}$	$2\frac{15}{16}$	$2\frac{5}{16}$	$1\frac{29}{32}$	$2\frac{3}{8}$	$1\frac{5}{8}$
NX	N	$3\frac{1}{2}$	$3\frac{1}{2}$	3	$3\frac{9}{16}$	$2\frac{15}{16}$	$2\frac{3}{8}$	3	$2\frac{1}{8}$

[1] For a closer figure, assume hole $\frac{1}{32}$ inch larger than bit.

The hammer drill has its best application in exploratory drilling underground, where relatively short holes are to be drilled, core recovery would be poor with a core drill, or core samples have no particular advantages over sludge samples. The hammer drill with special strong, independent rotation, using sectional steel, has been used to drill holes up to 272 feet in depth.[24] Hammer drilling has been discussed by Knaebel [25] and by Jackson and Knaebel [26] in earlier Bureau of Mines publications.

The advantages of drill exploration have sometimes been underrated in certain districts, whereas in others the reliability of ore-reserve estimates based on samples obtained by drilling has perhaps been overrated.

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[23] E. J. Longyear Co., Catalogue 8, pp. 59–60.

[24] Dobbel, Chas. A., Deep-Hole Prospecting at the Chief Consolidated Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677–689.

[25] Knaebel, John B., Sampling and Exploration by Means of Hammer Drills: Bureau of Mines Inf. Circ. 6594, 1932, 29 pp.

[26] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, 155 pp.

Drilling can often be employed to determine, at a fraction of the cost of underground exploration and in much less time, the position, outline, and approximate volume of ore in a lode or massive deposit; and, under favorable conditions, a close approximation to the average grade of ore may be made from drill samples. Drilling is particularly useful in determining various structural features, the position of faults, contacts between different types of rocks and shear zones, and obtaining other information of great use in planning underground exploratory workings.

Light, compact diamond-drill rigs and hammer drills are useful for drilling from underground stations in tracing the extension of veins or important structural conditions below the lowest level of the mine, and in probing for probable or possible blind lodes out in the walls of known deposits. By judicious use of drilling and interpretation of drilling results, considerable money often can be saved that otherwise would be wasted in drifting, crosscutting, and sinking.

In flat-lying bedded deposits or veins and ore bodies of considerable horizontal area, churn drills often can be employed to outline the deposits roughly, and, if the particles of valuable minerals are fine and uniformly distributed in the rock, churn-drill samples often will give very accurate information concerning the grade of the ore. Churn-drill samples may become contaminated with waste or salted if the hole caves, and allowance must be made for this in calculating the grade of ore from them or from sludge samples obtained in diamond drilling or hammer drilling. Moreover, in broken ground drilling water may carry away part of the cuttings into fissures and seams in the rock.

If the mineralization is erratic, less reliance can be placed on drill samples as a basis for estimating grade of ore than where mineralization is uniform. Even where ore occurrence is very erratic, however, drilling may be useful in determining quickly and cheaply the presence or absence of a vein or lode in a given zone, or of structural conditions favorable for the occurrence of ore.

Where a high percentage of core can be recovered, the core drill gives the most complete and most reliable information of any type of drilling. In precious-metal veins, where the valuable mineral is distributed very erratically, too much reliance should not be placed on sludge or even core-sample assays. The hole may pass through a particularly rich spot in a generally lean area or close to a rich zone in barren material. Results of drilling, therefore, must be checked by means of excavated workings, particularly where mineralization is erratic, before a high degree of accuracy is possible in estimating the average grade of the ore in this type of deposit.

The subject of drill exploration has been discussed by Jackson and Knaebel,[27] who give examples of practice and present data comparing the estimates of ore grades based on drill samples, with grades of ore actually mined later.

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[27] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, 155 pp. Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, 151 pp.

Table 6 is a compilation of data on cost of churn drilling, table 7 on costs of diamond drilling, and table 8 on costs of hammer drilling.

Drilling costs vary with conditions, such as character of the rock formations drilled, accessibility of drill stations, depth of the individual holes, wage rates and cost of equipment and supplies laid down at the operation, unit cost of power, kind of power available, and water supply. Under like conditions, the cost per foot of hole will be less if a large total footage is to be drilled (thus distributing installation and organization costs over more feet of hole) than for a small total footage, and, provided moving from one set-up to the next is not too expensive, will be less for holes of moderate depth than for deep holes.

TABLE 6.—*Examples of churn-drilling costs*

District	Material drilled	Size of hole, inches	Approximate depth of holes, feet	Cost per foot
Tri-State lead and zinc	Shale, sandstone, limestone, chert, flint.	5⅞ and 6¼	250	$1.00 to $1.25.[1]
Do	do	6¼	300	$1.00 to $1.50.[1]
Do	do	6¼	300	$1.00.[1]
Do	do	6¼	190	$1.10 to $1.50 in rock; $0.40 to $0.90 in shale.[1]
Bingham, Utah	Porphyry	23	1,469	$20.49.
Chino, N. Mex	do	13 to ?	945	$3.72.
Do	do	13 to ?	965	$2.75.
Bisbee, Ariz.[2]	do	------------	400	$1.97.[3]
Morenci, Ariz.[4]	do	10¾ to 4⅞	627 av., 906 max.	$3.257.[5]
Nye County, Nev.[6]	Clay and crystalline gaylusite.	5½	770 to 906	$1.033.[7]
Ludlow, Calif.[8]	Porphyry	8¼ to 6	258 av.	$1.043.
Miami, Ariz.[9]	Schist and porphyry	10 to 4¼	605 av.	$3.26.[10]
Burro Mountain, N. Mex.[11]	Porphyry	10 to 4¼	545 av.	$2.177.
Silver Bell, Ariz.[12]	Rhyolite, porphyry, and granite.	7⅝ to 4¼	230 av.	$2.52.[13]

[1] Contract price.

[2] Notman, Arthur, Churn-Drilling Costs, Sacramento Hill: Trans. Am. Inst. Min. and Met. Eng., vol. 52, 1916, pp. 444–457.

[3] 60,000 feet of drilling. Initial cost, $0.17; operating cost, $1.43; road building, $0.37.

[4] Grunow, William R., Churn-Drill Prospecting at Morenci, Ariz.: Eng. and Min. Jour., vol. 101, No. 23, June 1916, pp. 969–974.

[5] Drilling labor, $0.585; supplies, $0.453; tool sharpening, $0.043; power, $0.201; repairs, $0.140; total operating, $1.422. Sampling and assaying, $0.396; general expense, $0.230; preparatory expense, $0.973; geology and engineering, $0.018; first cost of equipment, $0.218, total cost, $3.257.

[6] Free, E. E., Drilling Costs in Potash Prospecting: Eng. and Min. Jour., vol. 100, No. 3, July 17, 1915, pp. 108–109.

[7] Drilling only.

[8] Palmer, C. H., Jr., Churn-Drilling Costs: Eng. and Min. Jour., vol. 99, No. 1, Jan. 2, 1915, p. 20. (24 holes totaling 6,456 feet.)

[9] Bowen, H. P., Churn-Drill Costs at Miami: Eng. and Min. Jour., vol. 99, No. 1, Jan. 2, 1915, p. 21. (73 holes, totaling 44,500 feet.)

[10] Drilling only, $2.75; roads, $0.44; water, $0.04; repairing, $0.03.

[11] Stauber, I. J., Churn Drilling in New Mexico: Eng. and Min. Jour., vol. 94, No. 11, Sept. 14, 1912, pp. 501–503. (57 holes totaling 31,100 feet.)

[12] Gentry, Martin Butler, Details of Churn-Drill Operations at Silverbell, Arizona: Eng. and Min. Jour., vol. 90, Oct. 29, 1910, pp. 850–851.

[13] 62 holes drilled.

From certain of the data in table 6, the total costs of large churn drilling-exploration campaigns may be deduced. Thus, 60,000 feet of drilling at Bisbee, Ariz., at $1.97 per foot cost $118,200; at Miami, Ariz., 44,500 feet at $3.26 per foot totals $145,070; 57 holes totaling 31,100 feet at Burro Mountain, N. Mex., at $2.177 per foot cost $67,705; 62 holes totaling 14,260 feet at Silver Bell, Ariz., cost $35,935.

At Globe, Ariz., the Old Dominion Copper Mining & Smelting Co. developed and at one time employed an electrically driven underground churn drill.[28] Four holes 12 inches in diameter were sunk for ventilation, and seven holes, starting with 8¼-inch bits and finishing with 6¼-inch or smaller bits, were prospect holes. The four 12-inch holes totaled 637.5 feet and cost $4,775; the seven prospecting holes totaled 1,124 feet and cost $5,052, including all items of cost such as moving, preparing drilling stations, and initial cost of equipment. Out of a total of $9,827, $2,382 was initial cost of equipment; $876, the cost of alterations to equipment; and $492, the cost of additions to equipment, a total of $3,750.

The following table shows the costs of churn-drilling eight different tracts in the Tri-State lead and zinc district:[29]

Property no.	Acres	Holes drilled	Holes in ore	Average depth, feet	Cost of drilling
1	80	129	35	250	$35,000
2	104	129	52	279	53,980
3	320	162	49	326	66,015
4	105	120	43	305	37,482
5	160	219	84	330	79,497
6	60	90	-----	300	27,256
7	80	138	55	235	40,839
8	40	78	47	300	26,450

The figures in the foregoing tables indicate that although, on the average, diamond- and churn-drilling costs are higher than hammer-drilling costs, the costs incident to the different methods of drilling are, broadly, of the same general order, especially when each is employed under conditions favorable to it. It has been pointed out that each type of drill has its peculiar limitations and each its particular field and that in certain kinds of ground or with certain depths or angles of holes one or the other may not suit at all.

Moreover, for sampling purposes, the reliability of results obtained rather than the cost per foot of hole should be the first consideration. The cost tables have been presented partly to show the range in drilling costs but are particularly significant in showing that where there is a choice of methods, the spread between costs by the different methods need not necessarily be great, and that the use of a cheaper drilling method to reduce the cost per foot of drilling is not justified if reliability of results would be sacrificed thereby. Misleading samples will be far more costly in the long run than any additional cost required to obtain dependable samples.

COST OF EXPLORATION

In the preceding pages some unit costs of sinking, drifting, crosscutting, and drilling have been presented, together with a few costs covering entire churn-drilling campaigns. Few figures are available covering the total cost of exploring a property before reaching a decision to develop it for production. However, figures are available covering expenditures for exploration and development at a few properties during various stages of exploitation before the productive stage, and the following data are summarized from those figures.

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[28] Bjorge, Guy N., Underground Churn Drilling at Old Dominion: Eng. and Min. Jour., vol. 103, Apr. 7, 1917, pp. 574–576.

[29] Reigart, J. R., The Cost of Developing to the Operating Stage and Equipping a Small or Medium-Size Mine in the Tri-State Lead and Zinc District: Bureau of Mines Inf. Circ. 6591, 1932, p. 9.

TABLE 7.—*Examples of diamond-drilling costs*

District	Material drilled	Size of bit or core, inches	Approximate depth of holes, feet	Cost per foot
Mineville, N. Y.[1]	Magnetite ore in gneiss	1-inch core		$3.92.
Ducktown, Tenn.[2]	Schist, graywacke, massive sulfides.	do	Over 150	$2.50.
Fierro, N. Mex.[3]	Magnetite, limestone		200 to 500	$1.441.[4]
Plumas County, Calif.[5]	Diorite and ore			$2.45–$2.70.[6]
British Columbia [7]	Chlorite, quartz-sericite schist, and ore.	⅞-inch core; 1 7/16 outside diameter.		$1.66–$2.46 direct; $1.77–$2.99 total.[8]
Southeast Missouri[9]	Limestone and shale		Average 600	$0.95–$1.40.
New Mexico [10]	Granite and diorite	⅞-inch core	Underground 250, surface 600.	$2.90. average.
Helena, Mont.[11]	Limestone and diorite			$3.00.
Rossland, B. C.[12]	Hard augite and diorite porphyry.	1 7/16-inch outside; ⅞-inch core.	Up to 1,000	$1.92.[13]
Alaska [14]	Diorite, quartz, greenstone, and slate.	1⅛-inch core		$1.743.[15]
Arizona [16]	Schist and porphyry	15/16-inch core; 1 9/16 outside diameter.	Average 208	$5.44.[17]
Sudbury, Ontario [18]	Norite, diabase, granite, greenstone.	⅞-inch core	Average 800	$2.53.[19]
New York [20]	Zinc ore, crystalline dolomite, gneiss.	do	Average 338	$2.05.
Mexico [21]	Ore and limestone	¾- and ⅞-inch cores	Up to 200	$0.506.[22]
New Jersey [23]	Hard, granitoid gneiss	1 3/32-inch core		$3.90.

[1] Cummings, A. M., Method and Cost of Mining Magnetite in the Mineville District, New York: Bureau of Mines Inf. Circ. 6092, 1928, 12 pp.; 31, 725 feet of drilling, 1918–19.

[2] McNaughton, C. H., Mining Methods of the Tennesee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[3] Kniffin, L. M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Bureau of Mines Inf. Circ. 6361, 1930, 20 pp.

[4] Drilling, $0.967; setting bits, $0.081; carbon, $0.207; moving, $0.186.

[5] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[6] Contract prices: Up to 400-foot holes, less than 15° inclination, $2.45; up to 400-foot holes, more than 15° inclination, $2.70 (75,378 total footage).

[7] Brennan, C. V., The Mining Operation at the Property of the Britannia Mining & Smelting Co., Ltd., Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6815, 1934, 36 pp.

[8] Higher costs for very siliceous ground.

[9] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

[10] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[11] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Bureau of Mines Inf. Circ. 6402, 1931, 11 pp.

[12] Couldrey, Paul S., and Sampson, E. H. S., Diamond Drilling as a Means of Intensive Development: Eng. and Min. Jour., vol. 110, No. 13, Sept. 25, 1920, pp. 608–610.

[13] 120,762 feet of underground drilling, 1902–1919.

[14] Eng. and Min. Jour., Diamond Drilling at Alaska Treadwell: Vol. 97, Jan. 17, 1914, p. 177.

[15] Labor, $0.798; carbons, $0.637; repairs, $0.012; supplies, $0.083; assaying, $0.002; power, $0.211; total, $1.743.

[16] Naething, Foster S., Diamond Drilling at Miami: Eng. and Min. Jour., vol 97, May 23, 1914, pp. 1039–1041.

[17] Based on 2,714 feet: Labor, $2.68; carbon, $0.82; miscellaneous supplies, $0.84; rent and depreciation, $1.10.

[18] Parsons, L. A., Diamond Drilling at Sudbury: Eng. and Min. Jour., vol. 102, Aug. 26, 1916, pp. 381–382.

[19] 100,000 feet of drilling: Labor, $0.90; carbon, $1.12; coal, $0.22; supplies, $0.04; depreciation, water, etc., $0.25; total, $2.53.

[20] Knaebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N. Y.: Bureau of Mines Inf. Circ. 6586, 1932, 25 pp.

[21] Walker, Harlan A., Mining Methods and Costs at El Potosi mine, El Potosi Mining Co., Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6804, 1934, 38 pp.

[22] Labor, $0.0887; carbon and bort, $0.1140; power, $0.0643; other direct costs, $0.2130; indirect costs, $0.0264; total, $0.5064.

[23] Sweet, J. R., Mining Methods and Costs at the Mount Hope mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp. (5,700 feet of underground drilling.)

NOTE.—Since table 7 was prepared there has been an appreciable reduction in diamond-drilling costs, and where a large amount of drilling is done costs of 50 cents to $1 per foot have become common.

TABLE 7.—*Examples of diamond-drilling costs*—Continued

District	Material drilled	Size of bit or core, inches	Approximate depth of holes, feet	Cost per foot
Ontario [24]	Schisted lavas and quartz porphyry, and vein material.	⅞- and ⅝-inch cores	Average 200±	\$0.72–\$0.87.[25]
Do.[26]	Schist and greenstone	⅞-inch core	Average 170	\$7.31.[27]
Do.[26]	do	do	Average 680	\$4.56.
Do.[28]	do	do		\$3.45.
Arizona [29]	Schist, diorite, greenstone, quartz, porphyry, bedded sediments, massive sulfides.	84 percent, ⅞- to 1⅛-inch core; 14 percent 1⅝-inch core; 2 percent 2⅛-inch core.		\$1.81 (1930).[30]

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[24] Young, Arthur W., Mining [Hollinger Consolidated Gold Mines, Ltd.]; Canadian Min. Jour., September 1935, pp. 395–396.

[25] Based on 166,654 feet of drilling in 1934, prices per foot (all costs included) vary with kind of rock, percent core recovery, and size of core, as follows:

Recovery	Porphyry	All other rock
⅞-inch core 90 percent		\$0.87
80–90 percent		.82
plus 75 percent	0.87	
70–80 percent		.77
65–70 percent	.82	
Minus 70 percent		.72
55–65 percent	.77	
Minus 55 percent	.72	

Price of ⅝-inch cores is 75 cents per foot, irrespective of core recovery.

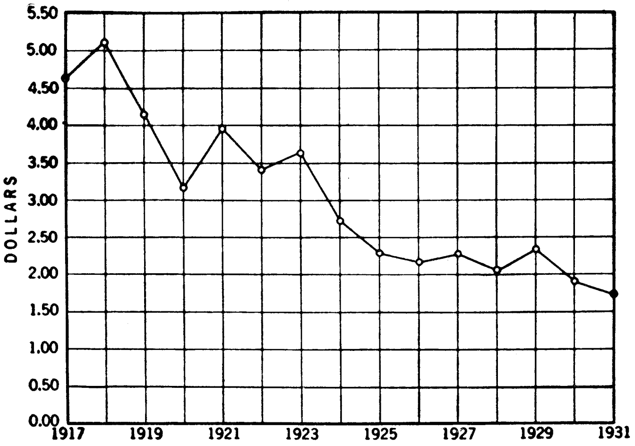
[26] Emens, W. H., and Jackson, Chas. F., Methods and Costs of Developing and Equipping the Ashley Gold Mine, Matachewan Gold District, Ontario: Bureau of Mines Inf. Circ. 6707, 1933, pp. 7-8.

[27] Surface drilling, 2,743 feet in badly fractured ground: Labor, \$2.61; carbon, \$3.20; other, \$1.50.

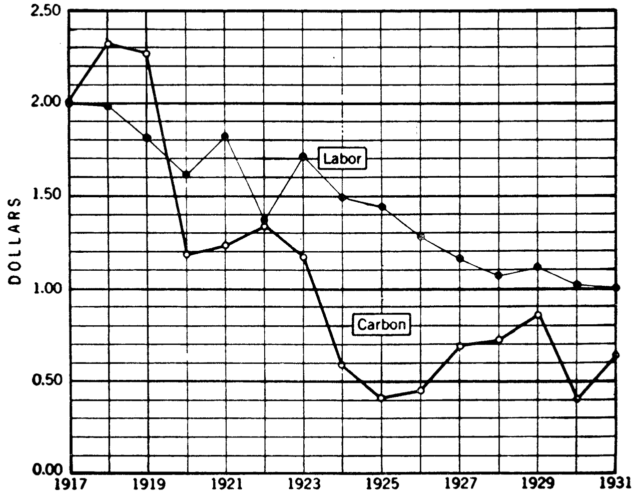
[28] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, p. 10.

[29] Hansen, Mayer G., Diamond Drilling at the United Verde Mine: Bureau of Mines Inf. Circ. 6708, 1933, 18 pp.

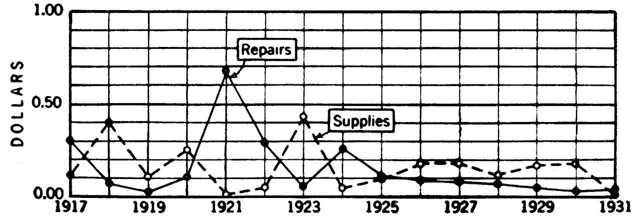
[30] Costs vary with depth of hole, kind of rock, and inclination of hole. See figure 3 for average costs in graphic form from 1917 to 1931.



A-Diamond drilling total cost per foot



B-Diamond drilling carbon and labor cost per foot



C-Diamond drilling supply and repair costs per foot

FIGURE 3.—Diamond-drilling costs, United Verde mine.

TABLE 8.—*Examples of hammer-drilling costs*

District	Material drilled	Depth of holes, feet	Size of holes, inches	Cost per foot	Remarks
Ducktown, Tenn.[1]	Schist, graywacke, massive sulfide.	Up to 120	3¾ to 2¼	$0.80	Total including shop and steel.
Tri-State, Kans.[2]	Cherty limestone			.64	
Coeur d'Alene, Idaho.[3]	Quartzite and vein material.	46 av., 129 max.	3¼ starter	1.10	Based on 2,868 ft., labor only, $0.92.
Fierro, M. Mex.[4]	Limestone and magnetite.			2.82	Drilling, sharpening, moving, labor only, $1,217.
Pecos, N. Mex.[5]	Schist and diorite	50–100		1.98	Labor only, $1.26.
Tri-State, Okla.[6]	Chert and limestone	148 max		1.69	
Do	do			1.00	
Do	do			1.90	
Do	do			1.70	
Do	do	152 max		.60–.70	
New Idria, Calif.[7]	Shale, sandstone, serpentine.	99 av., 228 max.	3¼–1¾	.75	
Tintic, Utah [8]	Limestone and ore	272 max	3⅛ or 2½ starters.	.97	Labor only, $0.44.
Edwards, N. Y.[9]	Zinc ore in dolomite, schist, and gneiss.	43 av		.99	
Gilman, Colo.[10]	Limestone, shale quartzite, ore.	228 max		.828	January 1925 to June 1931, 36,988 feet drilled.
Do	do			.505	May 1931.

[1] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, pp. 10–11.

[2] Banks, Leon M., Mining Methods and Costs in the Waco District: Bureau of Mines Inf. Circ. 6150, 1929, p. 3.

[3] Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho; Bureau of Mines Inf. Circ. 6238, 1930, p. 3.

[4] Kniffin, Lloyd M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Bureau of Mines Inf. Circ. 6361, 1930, pp. 5–6.

[5] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico.: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[6] Netzeband, W. F., Prospecting with the Long-Hole Drill in the Tri-State Zinc-Lead District: Min. and Met., vol. 11, June 1930, pp. 295–296.

[7] Moorehead, W. R., Methods and Costs of Mining Quicksilver at the New Idria Mine, San Benito County, Calif.: Bureau of Mines Inf. Circ. 6462, 1931, pp. 3–4.

[8] Dobbel, Chas. A., Deep-Hole Prospecting at the Chief Consolidated Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677–689.

[9] Knaebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N. Y.: Bureau of Mines Inf. Circ. 6586, 1932, pp. 6–7.

[10] Lary, Howard N., Exploration Methods and Costs at a Colorado Zinc Mine: Eng. and Min. Jour., vol. 132, No. 8, Oct. 26, 1931, pp. 353–355.

NORTHERN CANADA

The following conclusion with respect to exploration costs in northern Canada gold districts is taken from Information Circular 6693: [30]

> * * * although conditions differ at individual properties so that no estimate can be made of the expenditure required to prove a given prospect, the figures do indicate that in embarking upon the development of a prospect in this region, one should be prepared to expend *at least* $250,000 to $300,000 in order to determine whether the property will justify additional expenditure for a mill. It may require the expenditure of only a fraction of this amount in some instances to reach the conclusion that the property will not make a producer. On the other hand, this amount or much more may be required to prove that there is or is not a mine.

1. Central Patricia mine.[31]—This property consisted of 18 claims in northern Ontario, 120 miles from the railroad by winter tractor route. At the end of the exploration campaign, 33,155 tons of ore had been proved, together with an estimated additional "possible"

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[30] Jackson, C. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[31] Keast, A. J., and Jackson, C. F., Method and Cost of Exploring, Equipping for Development, and Developing the Central Patricia Group of Claims, Northern Ontario: Bureau of Mines Inf. Cir. 6681, 1933, 13 pp.

12,000 tons. A vertical, three-compartment shaft had been sunk to a depth of 527 feet, four shaft stations cut, and 1,776 feet of drifting and 1,019 of crosscutting completed. Preliminary work included 6,000 feet of diamond drilling, 1,050 linear feet of surface trenching, hauling of equipment and supplies, and construction of camp and temporary mine buildings. Although much of this work was of service later in development for production, the cost may reasonably all be charged to exploration. The summarized costs follow:

Preliminary exploration (staking, surveying, trenching, and diamond drilling)		$60, 280
Cutting winter roads and clearing site		7, 057
Buildings, plant and underground equipment (including freight and haulage)		89, 854
Underground exploration and development:		
Shaft, 527 ft	$41, 321	
Four stations and pump sump	6, 827	
Drifting, 1,775.7 ft	37, 691	
Crosscutting, 1,019.3 ft	24, 553	
Late surface exploration	172	
Total underground		110, 564
Cost of closing mine, pulling pumps, etc		1, 360
Total, all expenditures charged		269, 115

In addition, $60,325 had been expended covering inventory at time of temporary shut-down, and not charged off on the books.

2. *Ashley mine.*[32]—This property was 42 miles by winter road from Elk Lake, Ontario, the end of the railroad. It embraced 23 claims covering 951.7 acres. Discovery was made in October 1930, and the following figures cover expenditures during various stages for exploration and development up to the time a mill was completed and ready for operation (August 31, 1932) but excluding the cost of the mill:

October 1930 to December 31, 1930:		
Buildings	$2, 428	
Equipment and supplies	377	
Cookery stores, freight and teaming, miscellaneous payroll	9, 195	
		$12, 000
Dec. 24, 1930, to Feb. 27, 1931:	*Feet*	
Diamond drilling	1, 314	
Later drilling	3, 473	
Total	4, 787	29, 369
Shaft sinking (shaft inclined 53°, timbered, 6 ft. 2 in. × 17 ft. 0 in. rock dimensions)		-------
Feb. 16, 1931, to May 1931, 68 ft. (costs not distributed)		-------
May 1931 to December 1931, 436 ft		51, 923
Lateral development to Aug. 15, 1932:		
4,308 ft. of drifting, slashing, and crosscutting		97, 102
638 ft. of raising		15, 682
230 ft. of stations and pockets		13, 536
Payroll for Jan. 15, 1931, to Aug. 31, 1932 (not included in above accounts)		70, 921
Camp and mine plant buildings and structures		62, 870
Plant equipment not included above		53, 476
Materials and supplies not included above		75, 775
General expense (insurance, travel, haulage, and freight)		47, 869
Total		530, 523

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[32] Emens, W. H., and Jackson, Chas. F., Methods and Costs of Developing and Equipping the Ashley Gold Mine, Matachewan Gold District, Ontario: Bureau of Mines Inf. Circ. 6707, 1933, 28 pp.

3. *Bobjo.*[33]—This property is in the Patricia district, northern Ontario, 120 miles from the railroad. In 1928 and 1929 it was partly explored by a two-compartment, vertical shaft (6½ by 12 feet) to a depth of 268 feet. Two shaft stations were cut and 1,786 feet of lateral development was completed. The summarized cost of this work was as follows:

Road and trail buildings	$2, 108
Trenching, test-pitting, and general exploration	5, 027
Camp buildings	10, 133
Housing mining plant	2, 627
Tools, materials, and supplies	19, 621
Camp and mine equipment	36, 276
2,000 feet of diamond drilling	10, 525
Shaft and two stations	27, 168
Lateral development, 1,786 ft	37, 667
Organization of company, secretary-treasurer, etc	17, 350
Engineering direction	6, 435
Total	174, 937
Credit high-grade ore shipped	10, 474
Net cost	164, 463

4. *San Antonio.*[34]—This property is in Manitoba remote from railroad transportation. Following are the summarized costs of exploration:

General exploration, trenching, test-pitting, and diamond drilling	$91, 827
Camp and plant construction	12, 144
Camp and plant equipment	22, 909
Mine equipment—drills, steel, cars, pumps, etc	3, 381
Shaft sinking, 556 ft	37, 723
Cutting shaft stations, 76 ft	1, 146
Drifts and crosscuts, 2,579 ft	50, 039
Equipment and supplies on hand, not charged in above accounts	25, 530
Total	244, 699

ARIZONA

1. *Golden Belt.*[35]—This property is in Yavapai County, Arizona, 15 miles from the railroad. Following is a summary of the exploration costs, including camp buildings, roads, mine equipment and supplies, 750 feet of open-cuts, approximately 598 feet of tunnels and drifts, 95 feet of raises, and 825 feet of 14° incline, and cost of stoping 2,370 tons of ore taken out during the period covered:

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[33] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[34] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[35] Minton, David C., Jr., Cost of Equipping and Developing a Small Gold Mine in the Bradshaw Mountains Quadrangle, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6735, 1933, 10 pp.

	Appraised value as taken over May 1, 1931	Disbursements to Apr. 1, 1932	Total
Camps:			
Buildings	$2,000.00	$500.00	$2,500.00
Equipment and supplies	2,755.00	752.00	3,507.00
Roads	100.00		100.00
Subtotals	4,855.00	1,252.00	6,107.00
Mine:			
Equipment and supplies	3,362.00	2,531.08	5,893.08
Development:			
Direct labor and repairs	6,876.00	6,320.15	13,196.15
Repairs of property		142.95	142.95
Power		795.06	795.06
Subtotals	10,238.00	9,789.24	20,027.24
Total exploration cost, except trucking and proportion of small overhead costs	15,093.00	11,041.24	26,134.24

SAMPLING OF ORE DEPOSITS

A brief summary of the methods employed and the costs of prospecting and exploration has been presented in the preceding pages. The primary object of such work, as already stated, is to find ore and furnish information upon which to base estimates of its quantity and grade.

Estimates of ore grade are based on the assays of samples obtained from excavations in the ore, from drill holes, or both. The accuracy of the estimates will depend on the care taken in procuring the samples and the judgment used in deciding upon the sample interval required, the accuracy in assaying, and the proper weighting of the individual assays in combining them for determining average grades of individual ore blocks, especially the treatment of erratic high values. These are all matters that require experience and judgment on the part of the estimator.

Natural conditions affect the degree of accuracy possible; thus, the regularity of the deposit and the size and distribution of the particles of valuable minerals will limit the accuracy obtainable with a given pattern of samples. Also, there are practical considerations, such as the amount of money available and the degree of refinement required. In large low-grade deposits, a fraction of 1 percent difference in valuable mineral may be enough to determine whether the material is ore or waste. On the other hand, in a high-grade deposit, although a difference of a few dollars per ton in grade may make a large difference in the total profit recoverable from a deposit, such a difference may be relatively unimportant if the objective of the estimate is merely to determine whether or not development operations are justified. However, if the sampling is to form a basis for determining the price to be paid for the purchase of the property, obviously a high degree of accuracy is necessary.

In any event, sampling is an important operation, as much may depend on the care with which it is done and the intelligence shown in interpreting assay returns. It is obvious that correct conclusions cannot be reached from misleading samples—that is, from samples not truly representative of the material sampled. It is not always easy to obtain representative samples, and often they are reliable only when taken systematically in large numbers, so that errors in individual samples become compensating and not cumulative Systematic sampling requires a procedure that will eliminate the human factor as far as possible.

A perfect sample would have the assay value of the block of ground from which it was taken. Since valuable minerals are distributed unevenly and are present in varying degrees of purity throughout the gangue material, a perfect sample can be obtained seldom if ever. Indeed, in erratic ores, assays of individual samples may vary widely from the actual assay value of even the small block they are supposed to represent. However, if each sample is taken mechanically in strict accordance with a system and the number of samples taken is large, the combined samples may and usually do approach much nearer the actual average ore than do individual samples to the different blocks of ore from which they are taken, the errors compensating, as above stated.

The questions to be decided in adopting a sampling procedure include the size of individual samples and the sample interval. A well-recognized principle is that the finer the size of the mineral particles (ore and gangue) and the more even the distribution of the ore-mineral particles in the matrix, the smaller may be the sample and the greater the space between samples. Conversely, the more irregular the distribution of the ore minerals and the larger the particles, the larger should be the individual sample (in weight per unit length of sample cut) and the closer the sample interval. The mineralization in some deposits is so erratic that only actual mill tests of relatively large bulk samples will give a reliable index to the grade of the ore.

The following discussion will cover, principally, the methods of taking small samples rather than bulk samples. It may be said at this point, however, that mill tests of large or bulk samples may sometimes be employed advantageously to check the results of small samples previously taken from selected blocks of ground. Thus, the mill tests may establish a correction factor that may be applied to assays of small samples to give more accurate results for the particular ore in question. Methods of sampling ore deposits may be classified as follows:

1. Drill sampling with (*a*) churn drills, (*b*) core drills, (*c*) hammer drills, and (*d*) fish-tail and other types.
2. Face sampling: (*a*) Pick or chip sampling, (*b*) regular channel or groove sampling.
3. Grab sampling of broken ore.
4. Bulk sampling.

Each of these methods has been discussed at some length by Jackson and Knaebel.[36]

SAMPLING WITH CHURN DRILLS

In churn drilling for sampling purposes, the cuttings from each unit length of hole are brought to the surface and constitute a sample of the formation at the interval of depth from which it is cut. When cable-tool equipment is used, the tools are raised to surface and cleaned of material adhering to them by washing it off into the hole. The bailer is then lowered and the cuttings are raised to surface, bailing being continued until the hole is clean.

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[36] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, 155 pp.

——————, Gold Mining and Milling in the United States and Canada; Current Practices and Costs: Bureau of Mines Bull. 363, 1932, pp. 44–48.

Jackson, Chas. F., Knaebel, John B., and Wright, C. A., Lead and Zinc Mining and Milling in the United States; Current Practices and Costs: Bureau of Mines Bull. 381, 1935, pp. 65–80.

When hollow rods are used instead of cable tools, as with the Mesabi-type rig, the water is pumped continuously through the rods, as in diamond drilling, the return water carrying the cuttings being caught in a series of tubs, barrels, or other suitable settling apparatus.

With cable tools, the hole usually is bailed at regular depth intervals until ore is reached. The cuttings are examined mineralogically, often being panned to separate the constituent minerals. Thus, a knowledge is gained of the different rock formations and the horizon at which each occurs. Information gathered from examination of the cuttings should be logged carefully on suitable forms. When rods are used, the same information is obtained by watching the sludge during drilling and taking occasional pannings for examination, especially whenever a change is noted in its appearance.

When ore is struck, it is usually desirable and often necessary to case the hole from surface to the top of the ore, setting the bottom of the casing firmly into the ore. This is done to avoid contamination of subsequent samples with material loosened from the upper part of the hole by natural caving, by the raising and lowering of drilling tools and bailer, or by the slap of the drilling cable or rods, as the case may be. After being cased, the hole is cleaned thoroughly, and as drilling proceeds samples are taken at regular intervals (usually 3, 4, or 5 feet) by successively drilling and removing the cuttings from corresponding depth intervals. Care must be taken to clean the hole thoroughly each time and to collect all the cuttings in the sample.

Erroneous samples may result from caving of the hole; from leaking of drilling water in dry ground through cracks and crevices, carrying sludge with it; from rock falling from upper sections of the hole, thus contaminating the sample by raising or lowering the assay value, depending on whether the contaminating material is higher or lower in grade than the portion of the hole being sampled; and from faulty handling and treatment of the sludge constituting the sample.

Any great increase or decrease in the weight or bulk of the sample per foot of hole usually can be detected quickly and indicates either that the hole is caving badly or that appreciable leakage has taken place. Lesser departures from average size of samples are not so easily explained or detected. A given size of bit will cut holes of slightly different diameters in different rocks, depending largely on the hardness of the rocks, and small variations in size of samples from equal footages need not, as a rule, be looked upon with suspicion.

A large increase in size can usually be attributed to caving at the sample horizon or to sloughing from above, and assay results may require correction. Thus, if a sample weighs or assays appreciably more or less than the samples next above and next below it, serious contamination may be suspected. The procedure will then depend on conditions and previous experience with other holes in the deposit. It may be satisfactory to average the assay value of the vitiated sample with that of the two adjacent samples and assign the average to it, or to disregard the assay entirely, assigning to the sample the average of the two adjacent samples if they appear to be accurate.

Sludge samples from the larger holes are very bulky and usually must be cut down and dried at the drill. Generally, the first cuts or splits are made on the wet sample when cable-tool equipment is used.

At one time the senior author was employed on a churn-drill sampling job at Ray, Ariz., where holes were drilled with cable-tool rigs, starting with an 8¼-inch bit and followed, successively, by 6¼- and 4¼-inch bits when it became necessary to case and recase the hole. The samples from the larger diameters were bulky, especially in soft ground, and each bailer was dumped into a sample box, from which the water and cuttings passed through a sample splitter of the Jones type (fig. 4). The cutters were arranged in three tiers in such a manner that 50, 25, or 12.5 percent or all of the sludge could be caught as desired.

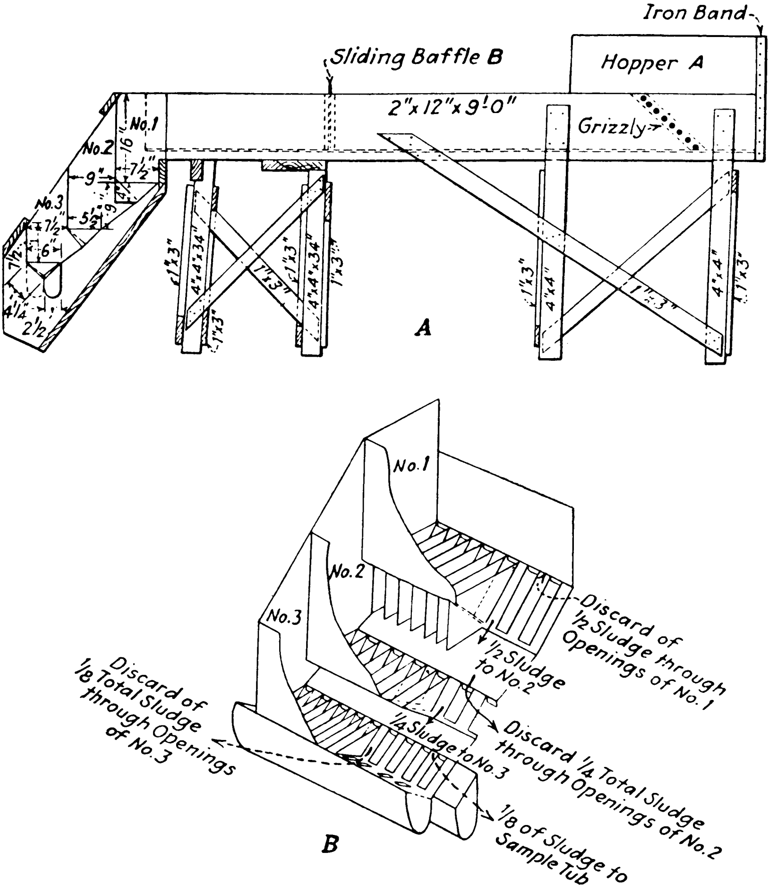
Iron Band
Sliding Baffle B
Hopper A
2"x 12"x 9'-0"
Grizzly
No.1
No.2
No.3
16"
7½"
9"
5½"
7½"
6"
4¼"
2½"
1"x3"
4"x4"x34"
4"x4"
A
No.1
No.2
No.3
Discard of ½ Sludge through Openings of No.1
½ Sludge to No.2
Discard ¼ Total Sludge through Openings of No.2
¼ Sludge to No.3
Discard of ⅛ Total Sludge through Openings of No.3
⅛ of Sludge to Sample Tub
B

FIGURE 4.—Churn-drill sample splitter. *A*, General assembly. *B*, Detail of divider, right side and a few partitions removed. (From Am. Inst. Min. and Met. Eng. Tech. Pub. 373.)

The final sample is then dried and sacked. In drying, it is important to avoid burning it or heating it to such a high temperature that chemical changes take place in the mineral constituents, particularly if sulfide minerals are present.

When churn-drilling is done with rods, the return water and sludge are discharged into suitable settlers, from which clear water overflows or is siphoned off, leaving the sludge in the settler. Whatever the arrangement used, it is important that the waste water shall not carry slimes away with it and thus remove fine material from the sample. This may be accomplished easily with some types of ore, but with those that contain clayey or other colloidal material it may be necessary to provide large capacity to allow a considerable time interval for settling.

ACCURACY OF CHURN-DRILL SAMPLES

In the Tri-State zinc and lead district, churn-drill samples have been used extensively for estimating tonnages and grades of ore in advance of actual mine development. The ore minerals are principally sphalerite and galena. According to Netzeband,[37] the experience of one large company in this district is that the ore mills out about 10 percent better than the estimates based on the churn-drill samples.

Keener [38] states, also, that in this district the estimates based on churn-drill samples are lower than actual mill-run tests.

Joralemon [39] states, with respect to cable-tool drill sampling of the disseminated-porphyry deposits, that "with careful work, involving casing below all caving ground, assays from churn drilling can be depended upon in nearly all cases to within 0.1 percent copper."

In referring to samples in the schist ore bodies at Miami, Ariz., Maclennan [40] has stated:

> The original churn-drill sampling was checked by (1) the usual channel samples, which were cut across the direction of the major seams every 5 feet of drifts; (2) cuttings from dry stoper-drill holes drilled across the major seams at 2.5-foot intervals; (3) samples of broken ore taken as the cars were loaded when driving the drifts; (4) a few check samples of 6 to 8 tons of ore shot down from the back of drifts over a length of 25 feet and carefully quartered down; (5) one 1,500-ton sample mined from a narrow shrinkage stope and put through an automatic sampler at the mill. This stope was sampled at the same time by the other methods. The conclusion reached was that the churn-drill samples were accurate, the stoper-drill samples the most accurate samples for small drifts, and the channel-sampling average 13 percent too high.

SAMPLING WITH CORE DRILLS

The advantages of obtaining core samples that provide a visual record of the rocks and ores sampled and at the same time furnish samples for assay have been pointed out under the caption Exploration by Drilling. The diamond drill, which employs either carbons or bort as cutting media, has been the type of core drill most widely employed in metal mining, largely because it will cut the hardest rocks; it can be used to drill at any angle from horizontal to vertical, up or down.

The shot drill has been used chiefly in obtaining cores of formations at dam sites, testing building foundations and the like, and sampling flat deposits of industrial minerals, where holes drilled vertically downward will provide the desired information. The shot drill has been used extensively together with churn and diamond drills for sampling copper deposits in northern Rhodesia.[41]

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[37] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Oklahoma; Bureau of Mines Inf. Circ. 6113, 1929, 11 pp.

[38] Keener, Oliver W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

[39] Joralemon, Ira B., Sampling and Estimating Disseminated Copper Deposits: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 611–612.

[40] Maclennan, F. W., Miami Copper Company Method of Mining Low-Grade Ore-Body: Am. Inst. Min. and Met. Eng. Tech. Pub. 314, 44 pp.; *also* 1930 Yearbook, p. 40.

With 100-percent core recovery the core will comprise an accurate sample of the ore penetrated. This does not imply that its assay value will be that of the ore at 5 feet or other distance from the hole; and, as with other types of samples, the results are reliable for estimation of ore grades only as cores are obtained at enough properly distributed points in the ore body to give an average value that approximates the average of the entire ore body. As the percentage of core recovery diminishes, the diameter of core varies, and other irregularities occur, the assay results become less reliable.

In soft or fractured formations, pieces of core are likely to break off and be ground by the sludge; the core often will be appreciably smaller in diameter in soft layers of rock than in hard layers, and therefore the core sample will contain proportionately less soft material than hard, whereas the sludge will contain more. If, as often happens, the softer material is appreciably higher or lower in grade than the harder material, the core assay will be correspondingly too low or too high. In soft or broken ground, a double core barrel usually will obtain a higher core recovery than a single core barrel.

In soft ground, the outside diameter of the hole may be larger than in hard ground, and the volume of the sludge sample will therefore be greater (assuming that all of it is caught).

When core recovery is low it usually becomes necessary to assay both the core and the sludge from each sample depth and to combine the results in order to approximate the actual value of the material cut by the drill. Mathematical methods based upon the relative volumes or weights of core and sludge and the assay values have been worked out for combining the results of core and sludge assays. In the opinion of the authors, some of these methods are more refined mathematically than usually is warranted by the approximations of some of the factors entering into the equations. Thus, the total overcut of the drill in soft ground would be determinable from the weight and volume of the sludge, but there is no way of determining whether the overcut is uniform over the full length of the sample or is principally from one small section that may be very much higher or lower in grade than the rest. Furthermore, there is no definite way of determining whether all the sludge has been recovered or whether some of it has been lost in cracks and crevices in the wall of the hole. If some has definitely been lost, the question remains as to whether the lost material carried a higher or lower percentage of valuable material than did the part recovered in the sample.

In diamond drilling, the drilling water is circulated down the drill rods, out the bottom, and up along the outside of the rods. In some recent models this direction of circulation has been reversed to impart maximum velocity to the rising current that carries the cuttings. This causes the sludge to be classified, the lighter particles rising readily to the surface with the water and the heavier particles lagging behind and perhaps coming to rest on a rough spot or in a crevice in the wall of the hole. With churn drilling, on the other hand, the sample becomes well mixed, and for this reason churn-drill sludge samples may be more reliable than core-drill sludge samples.

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[41] Matson, H. T., and Wallis, G. Allan, Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia; Trans. Am. Inst. Min. and Met. Eng., General Volume, 1931, pp. 66–82.

In view of the above uncertainties, which constitute only a few of the variables that must be considered, it is apparent that reliable interpretation of results depends on good judgment based on close observation of all abnormal occurrences in the taking of each sample, especially when only a small amount of core is recovered. A method of combining core and sludge assays that gives good results in one kind of ground might not apply to ground in which conditions are different. In ore of uniform grade and with even distribution of ore minerals throughout the gangue matrix, the sampling errors due to loss of core or caving obviously are not as serious as where mineralization is erratic and the grade varies widely from one point to another. Methods of combining core and sludge assays will be discussed briefly in a later section of this bulletin, but the foregoing has been introduced at this point to emphasize the necessity for care in collecting the sludge, especially where core recovery is poor, and in noting and logging everything unusual that may take place during the drilling and removal of each sample from the hole, so as to throw light upon the reliability of the assays later on.

Prior to drilling out of a sample the hole must be pumped clean of cuttings from previous runs and the depth at which drilling is resumed must be checked to make sure the bit is on the bottom of the hole. If the hole is caving, it may be necessary to case it to a point below the caving section. (If caving is anticipated, it is well to start with a large bit to permit casing and then to drill through the casing with a smaller bit.) It is then necessary to check the amount of water issuing from the hole with that being pumped in to determine whether an appreciable quantity of water is being lost. In fractured ground, very little of the water, or even none, may return, in which case it becomes necessary to cement the hole to close the fissures in the rock responsible for the loss of water. In Northern Rhodesia, holes were cemented if loss of water exceeded 5 percent.

When drilling is begun the return water is diverted to suitable boxes, launders, or barrels, where the cuttings are caught and the clear water is allowed to overflow. The simplest and perhaps most satisfactory device consists merely of a number of barrels, each provided with a number of bungs at different heights. As soon as the first barrel is filled, the sludge is diverted to the next, and so on. When all the cuttings have settled, the clear water is drawn off through the bungs and the sludge is collected in one sample for each run. A more elaborate arrangement, which was used with success in Northern Rhodesia, is shown in figure 5. This is suitable for surface drilling and deep holes where the cost of its installation is warranted. A simpler device and one widely used is a box 12 to 18 inches wide, 6 to 10 inches deep, and 4 to 10 feet long, divided transversely by two or more baffle plates, behind which the sludge settles and is caught. The discharge from the casinghead is diverted into the upper end of the box and the water overflows the lower end.

When the desired depth has been drilled the drill is stopped, and full pressure is put on the pump (when a pump is used instead of gravity head) to obtain as rapid circulation of water as possible in order to remove not only the light and fine material but the heaviest cuttings as well. The water is circulated until it comes up clear and free from cuttings. The depth to the bottom of the hole is checked on the drill rods and the core is broken off and pulled. The pieces of core are removed from the core barrel and arranged in a core box in the order in which they were taken from the hole. The top and bottom depths of the sample are then marked on the box opposite the corresponding ends of the cores. When core recovery is incomplete the length of the core will be less than the depth drilled. The pieces of core are measured along their centers and the sum of their lengths will approximate the length of core recovered. In solid, hard ground, the measured lengths may be accurate.

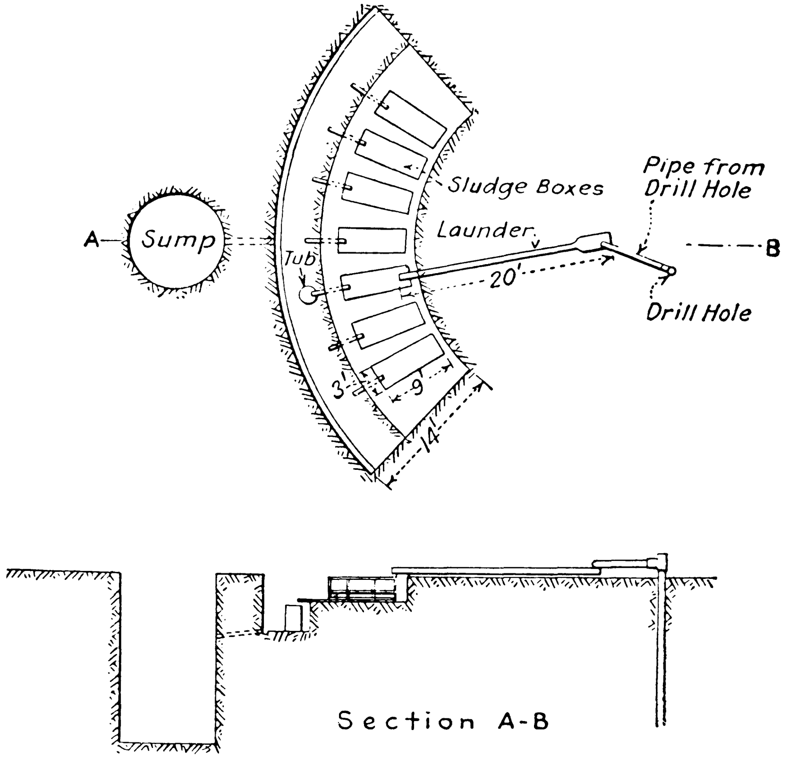

FIGURE 5.—Method of catching churn-drill sludge. (From Am. Inst. Min. and Met. Eng. Tech. Pub. 373.)

The cores should be inspected and the mineralogic and geologic information they reveal should be logged carefully by a competent geologist or engineer.

ACCURACY OF DIAMOND-DRILL SAMPLES

The foregoing makes it apparent that the accuracy of diamond-drill samples varies greatly and depends on a number of factors. Where the ore is fairly uniform in grade, core recovery is high, and cores are uniform in diameter core assays in general are very reliable. Where core recovery is low and assay results correspondingly doubtful the core obtained may nevertheless furnish valuable structural and other geological information. Where mineralization is erratic, as is true of most gold ores, core assays are often only indicative and are not trustworthy as a basis for estimating the grade of ore reserves.

Under most conditions, diamond-drill sampling is at least as accurate as other methods of drill sampling, and under conditions favoring use of the diamond drill, it is the most nearly accurate drilling method.

In the Southeastern Missouri district, where galena is finely disseminated in limestone but favors shaly layers and is much more friable than the limestone, diamond-drill samples are accurate. One company in this district assays only the sludge and retains the cores for reference, whereas another company bases its estimates on core assays and only assays the sludge as a check. Both report reliable results, which illustrates the point previously made that when the ore mineral is finely disseminated in the gangue and the ore is of low or moderately low grade, the reliability of samples is not seriously affected by disproportionate amounts of sludge and core, core and sludge assays being virtually the same.

SAMPLING WITH HAMMER DRILLS

The use of hammer drills for obtaining samples in underground work has increased in recent years. This method has advantages where holes are short, ground conditions preclude the use of diamond drills, satisfactory core recovery is impossible, or sludge samples are accurate enough for the purpose in view. Holes may be drilled at all angles from horizontal to vertical, up or down, but best results usually are obtained with holes drilled above the horizontal. The samples are small sludge samples, and previous statements concerning churn drilling are applicable here. Due to the smaller size of the holes in test drilling, the results of caving may be more pronounced than with churn drilling, especially as test holes seldom are cased.

Test-hole drilling with ordinary hammer drills is employed to check cut, channel, and grab samples or as a substitute therefor and is especially useful for obtaining samples of ground 5 to 20 or 25 feet beyond existing faces, particularly where ore boundaries are gradational or small seams of ore diverge from the main vein for short distances. Longer holes, up to over 250 feet, are drilled with heavy machines, as previously mentioned, as a substitute for diamond drilling where conditions are such that core recovery would be unsatisfactory.

The usual practice is to take a separate sample for each change in the formation or in the character of ore. These samples may be accurate unless lax methods are employed for collecting the sludge as it issues from the drill hole. If only structural information is desired, refinements in collecting the samples are not important; but if samples are desired for assay, it pays to take pains to assure collection of all the sludge and to prevent loss of slimes in the overflow from the collecting receptacle. The reliability of the samples will depend largely on the care taken in collecting the cuttings; and, as stated previously, it is better to take no samples at all than to be misled by faulty ones.

Drilling is done either wet or dry. When drilling is done dry, the samples may be caught in canvas bags fastened around the collar of the hole with the drill steel passing through an opening in the side of the bag. Open receptacles are sometimes employed, but loss of fine particles in the air may seriously affect the reliability of the samples.

When drilling is done wet, the samples are caught in boxes or cans placed on the ground below the hole, in small pans held below the collar of the hole by a helper, or in sludge boxes equipped with baffle plates for retention of the sludge. It is not always easy to place these receptacles so as to catch all the sludge, and with either small cans or boxes the overflow water may carry away slimes because of incomplete settling. By drilling a short hole with a large bit, sealing a pipe in the hole, and then drilling through the pipe with smaller bits, all the sludge usually can be caught and diverted to the settling device. A more elaborate method, which may be warranted when drilling deep holes, is shown in figure 6. As in churn drilling, it is important to provide ample settling time to collect the slimes with the sample, especially if there is an appreciable difference between the assay value of the fine and coarse cuttings, as often there is.

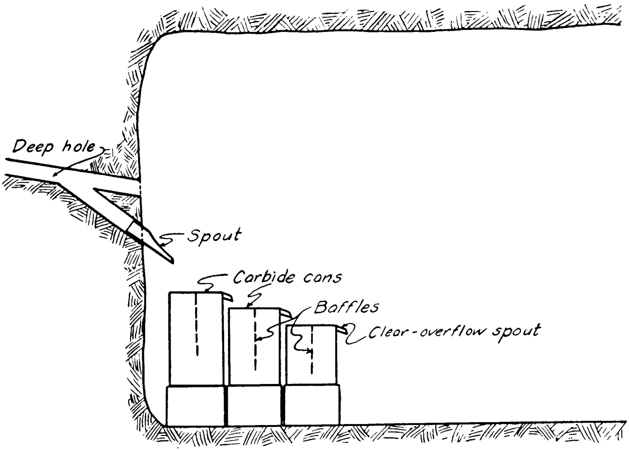

FIGURE 6.—Method of catching hammer-drill sludge.

ACCURACY OF HAMMER-DRILL SAMPLING

In this connection, some results reported by operators doing hammer-drill sampling are of interest.

At the Teck-Hughes mine,[42] Kirkland Lake, Ontario, 5-foot holes are drilled in the sides of the drifts at 10-foot intervals to test the walls and assist in keeping the faces in the best ore. The veins or lenses occur in shear zones in acid syenite, porphyry, and basic syenite, cut by parallel fault planes. The samples are accurate enough to be used in conjunction with cut samples in preparing assay plans upon which ore-reserve estimates are based.

At the Pilares mine[43] the ore mineral is principally chalcopyrite, which occurs as open-space filling in brecciated latite, andesite, and monzonite porphyry. The sides of drifts and raises are sampled by test-drilling with dry stoper drills, drilling holes 5 feet deep. The average grade of ore blocks is estimated by averaging the assays of the dry-stoper samples, and for several years the grade of ore delivered to the mill very closely approximated the estimated grade.

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[42] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6322, 1930, pp. 2–3.

[43] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares, Sonora, Mexico: Bureau of Mines Inf. Circ. 6307, 1930, pp. 6–7.

Maclennan[44] states that at Miami, where the copper minerals occur finely disseminated in schist, "stoper-drill samples [are] the most accurate samples for drifts."

At Fierro, N. Mex.,[45] deep-hole prospect drilling was used for a time, but the cost was higher than for diamond drilling and the information obtained was not as reliable. Holes up to 20 feet long continue to be used in stopes, raises, and drifts for exploration and sampling purposes, and the grade of the ore cut is estimated from the samples. The iron ore occurs as a replacement of limestone beds.

The long-hole drill has been used for underground prospecting in the Tri-State district for several years. The practice in this district has been described by Netzeband.[46]

The zinc and lead minerals occur in zones of shearing, shattering, and brecciation in limestone beds, particularly where there has been considerable silicification of the original beds. Drilling is done wet, and the cuttings are diverted into a powder box by a spout. The experience in this district, according to Netzeband, has been that no great reliance can be placed on assays of the cuttings, though this may be due in part to the lack of close supervision of the sampling by competent observers.

At Gilman, Colo., drill cuttings from each 6 feet of hole are caught in a powder box,[47] from which a sample of the sludge is cut from top to bottom. The ore is of a grade that does not necessitate close sampling and is so friable that the sample is usually contaminated. A composite assay of the sludge from an entire hole is regarded as accurate enough, however, to be used in ore-reserve calculations.

At Edwards, N. Y., prospect drilling with hammer machines is a regular feature of routine exploration work.[48] Here, zinc ore occurs as a sulfide replacement of silicified, folded dolomite beds. The sludge from the holes is assayed, but, since the ore minerals are very brittle, the samples are salted and generally assay about 25 percent too high.

In general, it may be said that where the ore minerals are not friable and have about the same toughness and hardness as the gangue or are disseminated in small particles uniformly through the matrix, hammer-drill samples may be accurate if precautions are taken to catch and settle all the sludge. If the ore minerals are brittle, occur in layers or bands, and are of very different hardness than that of the gangue, assay results will be high, due to salting. In such cases, an inch of rich ore cutting across the hole may salt several feet of sample or even the whole sample beyond the rich section. Even in such instances, however, samples may be valuable as indicative of mineralized formation if conditions are understood and too much reliance is not placed on the assay returns.

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[44] Maclennan, F. W., work cited in footnote 40 (p. 39).

[45] Kniffin, Lloyd M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Bureau of Mines Inf. Circ. 6361, 1930, pp. 5–6.

[46] Netzeband, W. F., Prospecting with the Long-Hole Drill in the Tri-State Zinc-Lead District: Min. and Met., vol. 11, June 1930, pp. 295–296.

[47] Lary, Howard N., Exploration Methods and Costs at a Colorado Zinc Mine: Eng. and Min. Jour., vol. 132, No. 8, Oct. 26, 1931, pp. 353–355.

[48] Knaebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N. Y.: Bureau of Mines Inf. Circ. 6586, 1932, pp. 6–7.

FACE SAMPLING

Face sampling is the cutting of pieces of ore and rock from exposed faces of ore and waste. The faces may be natural outcrops or faces exposed in surface trenches and pits or in the backs, walls, and floors of development openings and stopes. Face samples may be taken (*a*) by cutting grooves or channels of uniform width and depth across the face or sections of the face or (*b*) by picking off small pieces all over the face, more or less at random. The former method is the more systematic, and although ordinarily it is the one that is employed when accuracy is desired, pick sampling may sometimes be as accurate, or more so, depending on conditions.

Whichever method is used, the face should be properly prepared before actual taking of the samples is begun. Good preparation involves thorough cleaning of the face to remove adhering particles of dirt and ore minerals, and frequently some smoothing off is required to remove loose slivers and lumps or sharp projections of solid rock and ore. Washing with water under pressure generally will remove adhering particles, but in some instances this may have to be supplemented by scrubbing with a stiff brush, particularly if the face is a floor.

In channel sampling, the object is to cut a channel of uniform width and depth across the face in order to obtain in the samples equal weights of material from equal lengths cut. This objective can only be approximated in actual practice, but when a large number of samples is taken, errors usually are compensating if each sample is so taken as to approach the ideal as nearly as possible.

The weight required from each inch or foot of channel will depend on the uniformity of mineralization, the size of the mineral particles, and the richness of the ore; the less erratic the mineralization, the smaller the particle size, and the lower the grade the smaller the weight necessary to give a satisfactory sample. The length of each sample cut is carefully measured and recorded. Usual practice is to take about 1 to 2 pounds per linear foot of cut, although in very hard ground it may be difficult to reach this minimum. Sometimes considerably larger weights are required to obtain reliable samples.

In hard ground channel samples usually are cut by hand with hammer and moil, but sometimes with moils fitted to a light hammer drill. In soft ground they may be taken with a sample pick. The channel should be marked on the face with a crayon or lamp black before starting to cut the sample.

Various devices have been employed to catch the cuttings, such as bags held open by a metal ring and fitted with a handle, but the usual practice is to spread a tightly woven, smooth-faced square of canvas or other material on the ground below the point being sampled. In cutting samples from hard ground, it is often difficult to catch flying chips in this manner, and material belonging in the sample may be lost unless great care is exercised. Obviously, this loss may be reduced by using larger squares of canvas.

In channeling bedded or banded deposits, the long dimension of the channel cuts should be at right angles to the banding (see fig. 7, *A*), and usually it is desirable to sample each band or bed separately unless the hardness and the mineralization are uniform. If there are bands of decidedly different hardness or toughness, the material from each band should constitute a separate sample, because it is practically impossible to obtain equal weights per unit of length from each band. With erratic mineralization, it is usually best to limit the lengths of individual samples to 4 or 5 feet. In sampling gold ores, the usual practice is to exclude all particles of visible gold from the sample.

In sampling drifts, crosscuts, and raises, it is common practice to cut a channel sample or set of channel samples at regular intervals of 5, 10, or 25 feet along the opening, depending on the uniformity of the ore. A regular measured interval between sampling points should be used to eliminate the human equation in selecting them. Incidentally, a regular interval simplifies calculation of the grade of ore reserves later on.

Pick sampling consists of chipping off pieces from points well-scattered over the face. The ideal method would be to divide the face into measured squares and take a chip from each common corner, but such a refinement probably is rarely justified since in any event it is very difficult to obtain equal weights of material from the different points. In banded faces, pieces may be cut at points as shown in figure 7, *B*, the chips from each band constituting a separate sample.

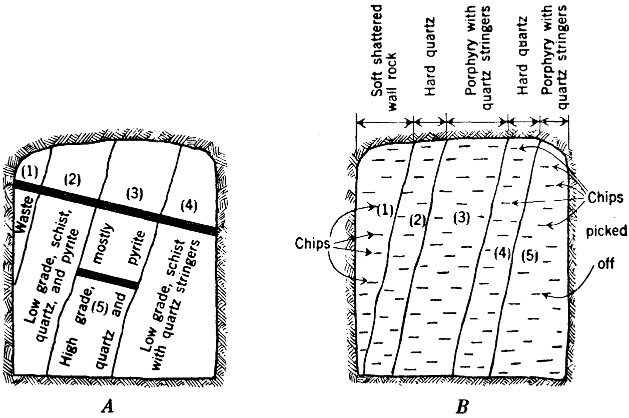
(1)
(2)
(3)
(4)
(5)
Waste
Low grade, schist, quartz, and pyrite
High grade, mostly quartz and pyrite
Low grade, schist with quartz stringers
A
Soft shattered wall rock
Hard quartz
Porphyry with quartz stringers
Hard quartz
Porphyry with quartz stringers
Chips
Chips picked off
B

FIGURE 7.—Examples of face sampling. *A*, Channel sampling; numbers indicate separate samples 3 and 5 averaged for high-grade band. *B*, Pick ("chip") sampling; numbers indicate separate samples.

Although chip sampling is not usually considered as scientific and accurate as channel sampling, it may give very reliable results in some ores.

ACCURACY OF FACE SAMPLES

Channel sampling probably is the method most widely employed in mine-examination work, and it may be inferred from this that, broadly speaking, experience has demonstrated it to be the most accurate under the widest range of conditions. There are exceptions to this, however, as at Miami, where it has already been noted that channel samples return assays that average 13 percent too high, whereas stoper-drill samples are more accurate.

A few typical examples of face-sampling practice and the results obtained are summarized below.

At the Hollinger mine, Timmins, Ontario, the grade of ore reserves is estimated principally from assays of channel samples.[49] The gold ore occurs in vein systems related to definite fractures within a main shear zone in schisted lavas intruded by masses of quartz porphyry. A crew of 45 men in charge of a head sampler is employed in underground sampling. A sampler will cut an average of 40 to 45 feet of channel in a shift. Routine channel samples are cut from all development and stope faces after each round, with check samples across the backs of development headings and slashes when required. The samples are cut with hammer and moil and are caught on canvas; about 1 pound of sample is obtained per linear foot of channel. Samples are cut normal to the banding; and each separate band, whether it be ore or waste, is sampled separately, its length being measured for proper weighting when results are combined. Overbreaks into waste in the stopes are sampled likewise, to provide a basis for estimating the average grade of all rock broken. All erratic high assays are cut arbitrarily to $50 before they are entered in the calculations. A high degree of precision is indicated by the experience over a long period of years, as shown by the following figures:

Tons milled	27, 500, 000
Value as estimated from sampling	$208, 071, 740
Value actually in mill feed	$206, 465, 707
Difference:	
Total	$1, 606, 033
Per ton	$0. 06

Channel samples from development headings average 12 to 20 percent higher than the ore actually stoped, but this is due to dilution in stoping, the fact that the drifts usually are kept in the best ore as far as possible, and to the deliberate mining of some ore of marginal grade.

At the Teck-Hughes mine in the Kirkland Lake district, Ontario, all development faces, drift backs and crosscuts, and stope breasts and backs are channel-sampled. High-grade streaks and bands of different hardness are sampled separately, as are bands of waste. Test holes are drilled into the walls, and the sludges from 2-foot sections are collected and assayed. The channel samples over a large area have averaged about 10 percent lower than the actual value. Visible gold is rejected from the samples before assaying.

At the Wright-Hargreaves mine in the Kirkland Lake district all faces are pick-sampled after each round is blasted.[50] Pick samples are taken (fig. 7, *B*) across each band, the material from each band constituting a separate sample. The width of each band is measured normal to its dip, and the widths are used for weighting the values in estimating the average grade of the face. This method has been found to check very closely with results from regular channel samples.

Bradley has given some results of moil samples of gold ores from the Alaska-Juneau mine in Alaska.[51]

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[49] Ringsleben, William C., Geology [Hollinger Consolidated Gold Mines, Ltd.], Canadian Min. Jour., September 1935, pp. 364–372. Irving, Andrew Gordon, Mine Surveying [Hollinger Consolidated Gold Mines, Ltd.], Canadian Min. Jour., September 1935, pp. 402–403.

[50] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits:Bureau of Mines Bull. 356, 1932, pp. 80–81.

[51] Bradley, P. R., Estimation of Ore Reserves and Mining Methods in Alaska-Juneau Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 103–109.

	Length of sections, feet	Average assay of moil samples	Mill returns, free gold plus tails
Crosscut 1	571	$0. 499	$1. 012
Crosscut 2	784	. 943	. 997

Methods of sampling at the Lucky Tiger mine, Sonora, Mexico, have been discussed by Mishler and Budrow.[52] The vein occurs along fractures in volcanic rock and the ore is narrow (1.1 to 1.9 feet, average). Stoping width averaged 3.4 feet. The ore was a high-grade silver ore with some gold associated with sphalerite, galena, pyrite, chalcopyrite, tetrahedrite, and stromeyerite in a gangue of kaolinized or silicified rhyolite.

All drifts, raises, and winzes were sampled at 5-foot intervals, each sample across the back was divided into as many samples as there were varieties of ore and waste, and the exact width of each was noted. Backs of stopes were sampled in the same manner. Following are some results of sampling along the Tiger vein. Odd-numbered samples were on one side of the drift and even-numbered samples on the opposite side.

Level	Number of assays	Average of assays, ounces silver per ton		Odd and even numbers	Error,[1] percent
		Odd numbers	Even numbers		
7	428	35. 4	41. 4	38. 4	7. 8
8	368	34. 3	31. 9	33. 1	3. 6
9	288	36. 0	42. 3	39. 2	8. 0
10	232	32. 0	39. 7	35. 9	10. 6
Total	1, 316	34. 7	38. 6	36. 6	5. 3

[1] Assuming that the average, 36.6 ounces, was correct.

Over a period of 14 years, estimates of ore reserves based on sampling averaged 34.0 ounces silver per ton, whereas the ore mined during the same period averaged 37.1 ounces.

The following is abstracted from a discussion of sampling at the United Verde mine by Quayle.[53] Faces usually are sampled either by chipping with a pick or, less commonly, by cutting channels. The chips are taken in four lines across the face for the width of the sample. The average sample weighs about 4 pounds, and individual pieces have a maximum diameter of 1 inch. A sample covers a width of 5 feet, and the height averages 7 feet. A stope sample of a face after a 6-foot round has been blasted represents about 20 tons of ore.

Sampling errors vary with the type of ore. In massive sulfides, the ore mineralization is uniform, and the error is very low—the samples average about 2 percent too high. In schist and porphyry ores, the occurrence of the ore minerals is very erratic, and the sampling error has ranged from 8 to 20 percent. For 1928 the average error for all classes of ore between the stopes and the smelter was 5 percent.

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[52] Mishler, R. T., and Budrow, L. R., Methods of Mining and Ore Estimation at Lucky Tiger Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 469–484.

[53] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

The foregoing examples indicate roughly the degree of accuracy that may be achieved by face sampling. Unfortunately, the published data on the accuracy of this method of sampling are limited. There are many articles describing face-sampling practice, but in most instances the accuracy is not indicated either because the samples are used for stoping-control purposes or for direction of development work only, and waste faces, rock from which is mixed with ore going to the mill, are not sampled. Thus, the dilution in stoping is not measurable, and there is no check between the mine and mill sampling. In other words, the rock going to the mill is not the same as that sampled in the mine. Many base-metal ores are readily distinguishable from the waste material by their appearance, and no cut samples are taken in the stopes. Where ore and waste cannot be distinguished readily, as is the case with most gold ores, many oxidized base ores, and some pyritic copper ores, channel or pick sampling provides a very satisfactory method of stoping control and is employed widely for this purpose; but unless the waste faces as well as the ore faces broken in stoping are sampled, no check between mine and mill sampling is possible.

In calculating the grade of ore reserves, the engineer often must base his estimates solely or largely upon face-sampling of development openings and must make allowances for dilution in stoping and for other factors.

At the Ground Hog mine[54] channel samples are cut with hammer and moil 3 inches wide and about 1 inch deep across the vein at 5-foot intervals. The raises are not sampled. Experience indicates that estimates based on these samples are too high in grade and too low in tonnage, because the sublevels are driven close to the hanging wall, where the grade of the ore is highest, whereas in stoping considerable low-grade ore is mined from the footwall side. About 25 percent more ore is mined than is estimated, and the grade is correspondingly lower. The ore contains silver, copper, lead, and zinc.

At the Verde Central mine,[55] face samples are taken of all drifts and raises as they advance. Channels are cut across the vein at the back of the drift at 10-foot intervals, and raises are sampled in the same manner. Channel samples are taken by hand with moil and chisel or with a stoper machine. Dilution factors employed in estimating grade of ore depend on experience and usually are greater for narrow than for wide stopes.

In one stope 5 feet wide the factor was 20 percent, whereas in another 20 feet wide and having good walls it was 5 percent. In one 15 feet wide, wherein andesite dikes made up about 10 percent of the area, a factor of 25 percent was employed. Chalcopyrite is the chief valuable mineral, and it occurs with quartz and pyrite in lenses in a larger mineralized zone.

At the Vipond mine,[56] where gold occurs in quartz veins and irregular masses of mineralized schist impregnated with quartz, channel samples cut from each working face in development headings and stopes were used for control of stoping and direction of development, whereas grab samples from muck piles were used as a basis for ore-reserve estimates.

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[54] Richard, F. A., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Bureau of Mines Inf. Circ. 6377, 1930, p. 4.

[55] Dickson, Robert H., Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.: Bureau of Mines Inf. Circ. 6464, 1931, pp. 3-4.

[56] Dye, Robert E., Mining Practice and Costs of the Vipond Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, 11 pp.

At Elkoro,[57] development headings were sampled after each round by cutting channels across the faces or backs. In stopes, cuts comprising three to five samples were made across the entire stope at intervals of 5 feet throughout its length. The grade, as estimated from the development samples, averaged about 17 percent high, owing largely to dilution in stoping. The valuable metals are gold and silver, which occur in veins with sulfides, quartz, and adularia.

In general, face samples may be said to afford a method of sampling within the range of accuracy required for control of stoping operations and direction of development headings, and when carefully done give a basis for estimating grades of ore reserves when proper allowance is made for dilution in stoping and for other experience factors.

GRAB SAMPLING

Grab sampling usually consists in taking portions of ore at random from muck piles, chutes, or the tops of cars of ore. When done in this manner it is a haphazard method that can be relied upon only for roughly approximating the grade of the ore and often is employed merely for that purpose. The human factor or tendency to select for the sample too large a proportion of the best-looking pieces of ore, pieces of convenient size, or only the fines is more inherent in the application of this method than in other methods of sampling. Data on results of grab sampling at many mines indicate that usually they are too high.

The authors are of the opinion, however, that if grab sampling is done as systematically and with the same care commonly exercised in channel sampling, equally reliable results may be obtained, especially with ores of certain types. There are some ores in which the valuable minerals are contained almost entirely in rich patches or bunches in almost barren rock, like plums in a pudding, and that cannot be sampled accurately by channeling but that can be sampled with a fair degree of accuracy when broken by taking grabs from many points to form large samples.

As with other methods of sampling, the degree of refinement warranted by the purpose and cost of sampling should be considered.

The errors inherent in haphazard grab-sampling may be reduced by taking large samples—1 shovelful out of every 5 to 10 (or other proportion, depending on the nature of the ore and size of the individual pieces)—during loading into cars, or by using a scoop and taking one scoopful of ore from points equally spaced, by actual measurement, over the top of the pile (fig. 8). A convenient way to spot the points on a muck pile is to use a rope with knots or metal tabs at 1-foot intervals and stretch the rope across the pile on parallel and equally spaced lines. At each knot or tab, one scoopful should be taken for the sample. If the knot falls on a lump larger than the scoop will hold, a piece equal to a scoopful should be broken off under the knot with a hammer. This method may also be used for sampling loaded cars of ore, and when so employed samples should be more reliable when taken from the freshly loaded car before it has been trammed or hauled a long distance with resultant settling of the fines.

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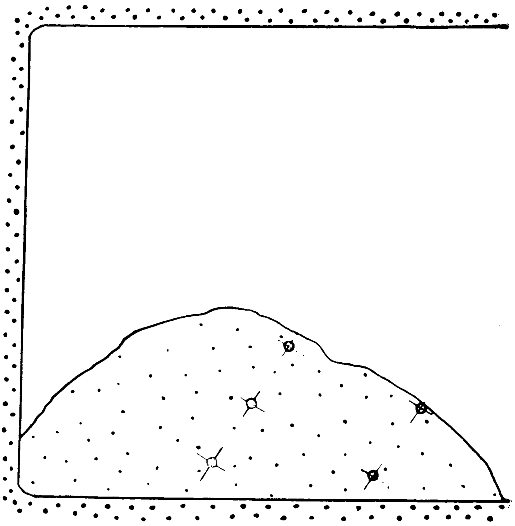
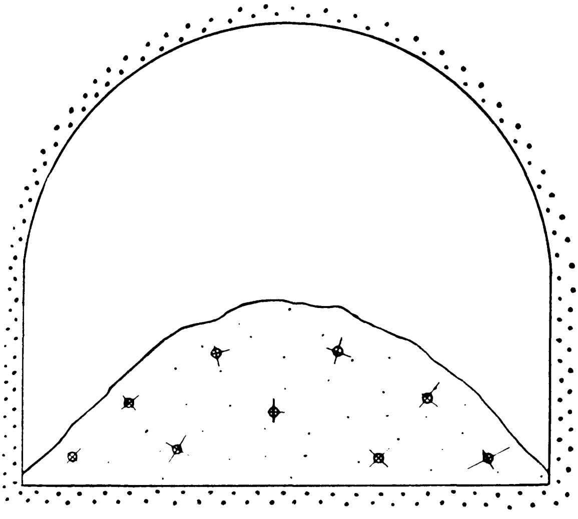
[57] Park, John Furness, Mining Practices, Methods, and Costs at Elkoro Mines, Jarbidge, Nev.: Bureau of Mines Inf. Circ. 6543, 1931, 12 pp.



Long. section



Cross section

Figure 8.—Grab sampling, showing equal spacing of sampling points on muck pile.

Either of these methods largely eliminates the human factor in grab-sampling by making it systematic, and although the samples of individual piles or lots of ore may be high or low, errors will become compensating as more and more lots are sampled in the same systematic manner.

ACCURACY OF GRAB SAMPLES

Grab-sampling usually is employed where a high degree of accuracy is not expected, for control of breaking in stoping and drifting, as a check against channel or drill-hole samples, for determining the approximate grade of ore from individual stopes or chutes, or as a general check on the grade of mill heads.

At the Vipond mine, Timmins, Ontario, only [58] a very small proportion of the gold is visible, by far the greater part being finely disseminated throughout the mass and favoring the pyrite, quartz, and schist in the order named. Diamond-drilling, channel-sampling of development faces and backs, stope faces and backs, and grab-sampling of the broken ore at each working place (stope or drift) all were employed. Grab samples consisted of a handful of fines from each car as it was loaded, and although this method is crude and day-to-day samples varied considerably, the results obtained were much more valuable in regulating the mill feed and indicating the grade of ore currently being obtained from various working places. The value of the ore milled during 1930, as calculated from the muck-(grab-) sample assays, was 4.8 percent below actual mill recovery plus tailing loss.

At the Wright-Hargreaves mine in the Kirkland Lake (Ontario) gold district, grab samples from muck piles and cars ran 30 to 40 percent high.[59]

At the Douglas Island mines (Alaska), Bradley [60] has summarized the experience with grab samples over a period of years as follows:

	Average assay of muck samples	Recovery plus tailing
Treadwell mine (17 years)	$2. 37	$2. 47
700 mine (14 years)	2. 20	2. 33
Mexican mine (16 years)	2. 88	2. 83
Ready Bullion mine (15 years)	2. 72	2. 24

Jones [61] has stated that in the Cripple Creek (Colo.) district grab samples taken from chutes (a handful from each car trammed) run 20 percent higher in value than the settlement value of the ore.

In the Mogollon district, New Mexico, gold-silver ore occurs in veins in a gangue of quartz, calcite, wall-rock fragments, and some fluorite. According to Kidder [62], a grab or "scale" sample was taken from each car as the ore was trammed to the mill and a composite sample made for each shift. At the end of the month the calculated average of scale samples was checked against the chute samples taken underground and against the average mill heads (calculated from production plus tails). On account of the slightly higher assay value of the fines in the ore, the average mine and scale samples usually were slightly higher than the mill-head sample. The annual average, however, seldom showed a variation of more than 10 percent, compared with the mill heads, and generally agreed within 3 or 4 percent.

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[58] Dye, Robert E., Mining Practice and Costs of the Vipond Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, pp. 3–4.

[59] Jackson, Chas. F., and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, pp. 80–81.

[60] Bradley, P. R., Estimation of Ore Reserves and Mining Methods in Alaska Juneau Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, p. 104.

[61] Jones, Fred, Mining Methods in the Cripple Creek District: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 512–517.

[62] Kidder, S. J., Mining Methods in Mogollon District, New Mexico: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 529–549.

Catron[63] states that at the Colorada mine, Cananea, Mexico, results from grab samples taken from cars as they reach the station check the returns from the mill and smelter sample plants closely enough for purposes of control of production.

At the United Verde mine,[64] samples of broken ore in the stopes or car or chute samples were necessary. It was determined that for such sampling, the proper-size pieces to choose were 3½ inches in all dimensions. Several cars of ore of approximately 1 ton each were chosen from various localities and sorted by hand into different sizes—minus 1 inch, plus 1 minus 3 inches, plus 3 minus 5 inches, plus 5 minus 7 inches, plus 7 minus 10 inches. Each size was weighted and assayed and the average assay determined for each car. It was found that the assay of the plus 3- minus 5-inch size was nearest that of the whole car, and the plus 1 minus 3-inch size second best, from which it was deduced that the proper size to choose must be slightly over 3 inches in diameter.

At the Eighty-Five mine, Valedon, N. Mex., mine samples (of copper ore) were generally 10 to 15 percent higher-grade than smelter samples.[65]

At the El Potosi mine, Chihuahua, Mexico,[66] experiments have proved conclusively that at this property the easiest and most efficacious method of sampling for general mining control is to take hand-grab samples from mine cars. The difficulties inherent in grab-sampling ores carrying appreciable lead values in the form of galena are recognized. Daily assays are made for silver, lead, and zinc, and the results are adjusted by predetermined factors for the three metals. The factors are obtained by comparing the weighted average of all assay returns during the month with the average mill heads, which are obtained from weightmeter and automatic samplers. Thus, comparisons may show mine-car samples to be 12 percent high in silver, 20 percent high in lead, and 5 percent low in zinc. For the following month the mill assayer applies these percentages as correction factors to the mine-car assays.

CUTTING DOWN SAMPLES

Before assaying, original samples must be cut down, often in several stages. Each cut involves, in effect, the sampling of a sample, the final portion comprising only the ore that actually goes into the crucible or flask and upon which the actual determinations are made.

In cutting down samples to reduce the bulk to the final amount required for assay, it is important to observe the general principles already mentioned regarding particle size, richness of the ore, and uniformity of mineral distribution in relation to the size of the sample and of the cuts made. Thus, it may be necessary to reduce the particle size at one or more stages to maintain the proper relationships.

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[63] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Bureau of Mines Inf. Circ. 6247, 1930, p. 6.

[64] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, p. 6.

[65] Youtz, Ralph B., Mining Methods at the Eighty-Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.: Bureau of Mines Inf. Circ. 6413, 1931, p. 5.

[66] Walker, Harlan A., Mining Methods and Costs at El Potosi Mine, El Potosi Mining Co., Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6804, 1934, pp. 10–11.

The required fineness before cutting at any stage depends on (*a*) size of sample, (*b*) distribution, size, and number of particles of ore mineral in the sample, (*c*) range in value between high- and low-grade ore mineral particles, and (*d*) percentage of the total weight of the sample comprised in the valuable mineral.

Tables have been devised by various investigators as a guide to the relationships that should be maintained; these have been based both on mathematical calculations and on experimental data.

The following table by Woodbridge,[67] based on extensive experiments and calculations by Brunton,[68] is an example.

TABLE 9.—*Smallest permissible weight of sample for varying sizes of crushing of medium-grade ore*

Maximum size of products, inches	Smallest permissible weight, pounds	Maximum size of products, mesh	Smallest permissible weight, pounds
2	10,000	6	50
1½	5,000	10	25
1	2,000	18	10
¾	1,000	30	4
⅜	400	50	1
¼	300		
3⁄16	100		
⅛	75		

This table was devised as a guide for sampling large lots of medium-grade gold ore, but the principle involved is the same as that for cutting down the kinds of samples under consideration. If the ore minerals are very unevenly distributed in the gangue matrix or the ore is high-grade, finer crushing or larger samples will be required. For more uniformly mineralized base ores, considerably smaller weights would be allowable.

The handling and treatment of drill samples have been touched upon briefly already. Sludge from drill holes usually is fine enough to permit splitting or cutting at the drill, and it has been noted that in churn-drill sampling at some mines the sludge is split in the wet state when it is bailed from the hole, sometimes only ½, ¼, or ⅛ of the total sample being retained.

Face samples often are sacked and sent directly to the assay office without being cut down. When prospects or mines remote from an assay office are sampled, it may be advisable or even necessary to reduce the bulk and weight of the samples before shipment.

Samples may be cut down either by hand or mechanically. Mechanical splitting is preferred for accuracy, but mechanical splitters may not be available and hand-cutting may be the only alternative. Large samples usually are first cut down by "quartering," in which process the ore should first be spread on a clean, smooth floor and pieces larger than allowable should be broken.

The sample is then shoveled into a conical pile, each shovelful being dropped on the tip of the cone. The fines will tend to remain near the tip or slide part way down the sides of the cone, whereas the larger lumps will roll to the bottom. Thus, the fines will tend to become more or less segregated from the lumps, which is not conducive to thorough mixing. In practice, the tip of the cone will tend to move in one direction, so that when the cone is completed the tip will be off center. If this occurs, there will obviously be a concentration of lumps on one side; when the cone is flattened and quartered, some quarters will have too large a proportion of lumps; and as the lumps often contain less or more of the valuable mineral than the fines, the quarters retained in the sample will not represent the average grade. Some samplers employ an upright iron rod to mark the center of the cone and thus avoid "drawing" the tip to one side. Others employ a wooden or steel cross having arms of equal length, which is laid on the floor, and each shovelful is then dropped over the intersection of the arms.

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[67] Woodbridge, T. R., Ore-Sampling Conditions in the West: Bureau of Mines Tech. Paper 86, 1916, p. 57.

[68] Brunton, D. W., The Theory and Practice of Ore-Sampling: Trans. Am. Inst. Min. Eng., vol. 25, 1896, p. 826.

After all the ore has been swept up and coned, the next step is to flatten the cone into a disk by dragging it out spirally with a shovel or flat blade, beginning at the tip of the cone and taking care to spread the material uniformly in all directions. The thickness of the finished disk should be about one-tenth its diameter, and, if flattened properly, it will be circular. It may be necessary to recone and flatten one or more times (three conings is common practice) to mix the sample thoroughly before quartering.

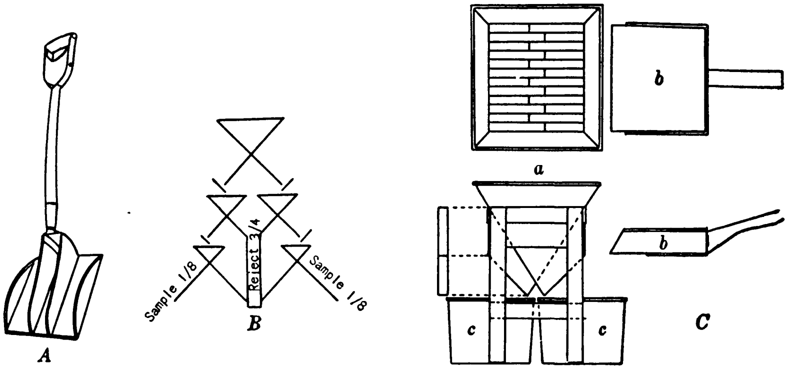

FIGURE 9.—Methods of splitting samples. *A*, Brunton quarter shovel. *B*, Diagram of bank or combination riffle sampler. *C*, Jones splitter.

The final disk is then divided into quadrants by marking with a straight-edge or a right-angle cross, and two opposite quadrants are removed and rejected. The two remaining quadrants constitute the sample. This may be reconed and requartered in the same manner as the original sample if still of sufficient bulk in proportion to the size of the particles, or it may be first crushed or ground to smaller size before quartering or splitting it further.

First cuts of original samples may be made also with such devices as the Brunton quarter shovel (fig. 9, *A*) or a split shovel. In splitting a sample with the Brunton shovel, the shovel is forced into the sample pile until all compartments are filled. The shovel is then removed and tilted backward to discharge the material in the two outside compartments, which is the reject. The portion in the middle compartment is emptied into the sample box. The operation is continued until the entire sample has been handled. The split shovel is constructed with alternate openings and prongs, the latter forming narrow boxes, the area of which equals that of the openings between them. The shovel is held over a box or other receptacle and an assistant shovels the sample over it. Half the material is supposed to pass through the openings into the box, the half remaining on the shovel being rejected.

A bank or combination riffle sampler like that shown in figure 9, *B*, can be constructed easily, and by shoveling the sample into the top, a ⅛-sample is cut. Figure 9, *C*, illustrates the Jones splitter, which is usually used for the last splitting after the sample has been ground to pass a ⅛-inch or finer screen. The final split for the portion that actually goes into the assay is made upon finely pulverized material that has been mixed on a sample cloth. The mixed sample is spread into a thin cake, and the portion for assay consists of dabs taken from points well distributed over the cake.

Automatic samplers of the revolving type, which cut small portions from a stream of ore at regular intervals, are employed extensively for sampling in ore-dressing plants. Samplers of this type give quite accurate results, provided they are kept in good working order and they are watched to see that they do not become clogged and that the sample receptacle does not fill and run over.

Whatever the system of cutting-down employed, the danger of error caused by inclusion of disproportionate amounts of valuable mineral particles, either in the reject or in the sample, increases as the size of the sample becomes smaller. By way of illustration, assume a sample cut in which only one particle of valuable mineral remains. If the particle goes into the reject during cutting-down, the final sample will have a zero value, whereas, if it goes into the final sample for assay, the result will be 2, 4, 8, or more times the correct value. Now if this particle is broken up into many smaller pieces by grinding the sample, the likelihood of obtaining a proportionate amount in the final sample is much greater. It therefore follows that the finer the particles comprising the sample at each stage of splitting, the better is the chance that the final portion used for assay will be truly representative.

In sampling gold ores it is customary to reject any visible gold. Particles of free gold are not pulverized during crushing and grinding but merely have their shape changed. Therefore, coarse particles of gold in the original sample may carry over into the final sample and give an assay value many times the actual value of the ore.

SUMMARY OF SAMPLING

The accuracy of the samples themselves—that is to say the extent to which they truly represent the material actually sampled—should not be confused with the accuracy of ore-reserve estimates based upon assay results. Samples may very accurately represent the grade of the ore sampled; but in mining, the grade of the ore sent to the mill may be considerably lower, due to dilution in stoping operations because of inadvertent inclusion of wall rock or because of deliberate mining of ore of lower grade than that sampled. The necessity for eliminating the human factor in taking samples has been stressed, and the advantage of making the sampling operations as automatic as possible for this purpose has been pointed out. By adhering to rigid standards for the spacing of samples and the method of taking them, size of samples, relationship of sample size to particle size in cutting down, the mathematical probability is greater that errors in individual samples will be compensating. Interpretation of sampling results in estimating grade of ore reserves, on the other hand, is a matter requiring the exercise of judgment.

ESTIMATION OF ORE RESERVES

Estimates of the tonnage and average grade of ore deposits (ore-reserve estimates) are made for various purposes. They may be made by examining engineers as a basis for placing a value on a mining property in connection with reports on behalf of owners or vendors, on the one hand, or for prospective purchasers or lessees, on the other. They are made for tax purposes and sometimes in connection with mergers of two or more companies or with litigation.

Most operating companies make periodical ore-reserve estimates, usually at least annually, to determine their ore-reserve position as a basis for controlling development and exploration and allocation of funds therefor; for determining deferred, depletion, and depreciation charges per ton; or as a basis for deciding upon operating policy—expansion or contraction of operations, capital expenditures, and the like.

The final estimate in any instance usually is a composite of estimates of different blocks or areas which often differ appreciably from each other in grade or character of ore. A continuous ore-reserve inventory, by blocks, levels, and stopes, in a mine may be required as a basis for controlling stoping operations to maintain a desired grade of output by mixing ores from different blocks.

Some mines produce two or more grades or kinds of ore, which must be mined, milled, and shipped separately; and it becomes necessary, then, to know the grade and type of ore available in each section of the mine.

Thus, iron ores are often mixed to give a desired iron, phosphorus, silica, manganese content. The copper-bearing sulfide ores of Ducktown, Tenn., chiefly valuable for their sulfur, are roasted for the manufacture of sulfuric acid; it is necessary to grade the ore from different stopes carefully, on the basis of sulfur content, to form the right mix for this purpose.

Ore-reserve estimates are based upon the results of exploration and development and analyses of the samples derived therefrom. Unless a deposit is fully developed (and even then to a lesser degree), certain assumptions have to be made regarding the continuity and grade of ore between exposed faces or drill holes that have been sampled. In making these assumptions, the engineer must interpret all available information, and the accuracy of the final results will depend largely on his experience and the soundness of his judgment. Not only must he correctly combine the assay values of the samples, but he must interpret the geological criteria and consider the influence of structural conditions on continuity and grade of ore, the probable loss of ore in mining, the dilution with waste (which, in turn, may depend on the mining method employed or to be employed), and the cost of mining and milling, which may be an important factor in determining the minimum grade of rock that can be classed as ore.

It is apparent, then, that estimation of tonnage and grade of ore reserves is not a precise science. In some districts where the character, uniformity, and habit of the ore bodies have been learned from long mining experience, more or less arbitrary methods may be applied with considerable confidence in estimating ore extending beyond exposed faces. Thus, in Minnesota and Michigan the mining companies and the State tax commissions accept valuations of ore reserves based upon certain rules as to continuity and grade of the ore.

Ore-reserve estimates include the determination of (1) tonnages of ore and (2) average grade or value per ton. Since the grade or content of valuable metal establishes the difference between rock that may and may not be classed as ore, tonnage cannot be estimated without considering the question of grade.

Consideration of grade in connection with tonnage estimates may or may not present a complex problem. Thus, in some deposits with sharply defined walls or boundaries between which all the material is known by experience or by actual development to be of ore grade without the necessity of averaging the sample assays, the tonnage can be estimated without particular reference to assays. In other types of ore bodies having no definite boundaries and with values gradually petering out, the limits of minable ore have to be determined by assay; hence, tonnage and grade estimates are interdependent. With large low-grade deposits that can be mined profitably only by use of large-scale nonselective methods, it may be necessary to compute tonnages and grades on the basis of several different assumptions as to the mining limits. The first estimate may indicate that too much marginal ore has been included to give an average grade that will leave the required profit margin, and it then becomes necessary to establish new hypothetical ore boundaries and reestimate the average grade of the corresponding tonnage.

This process may require repetition by cut-and-try methods before a final acceptable estimate is reached. It involves a determination of the minimum grade of material that can be included, but the mining method to be employed may preclude the possibility of eliminating all material below this grade, hence the necessity for cut-and-try estimates in such instances.

Some deposits are so uniform in grade that satisfactory ore-reserve estimates can be made by merely computing the volume and tonnage of ore and assigning to it the grade of the ore produced during the preceding year or other period. Except for such deposits, the initial step in making ore-reserve estimates is the preparation of assay maps and sections. These not only furnish the data necessary for actual measurements and computations, but provide a general picture of the ore bodies, the variations in grade within them and along their boundaries, and thus the nature of the irregularities of mineralization.

In making ore-reserve estimates as a basis for valuing a property for sale or other purpose, most engineers, to avoid overvaluation and at the same time recognize potentialities for ore as yet undisclosed, estimate tonnages under three separate headings: "Developed ore" (sometimes termed "positive ore"), "probable ore," and "possible ore." The judgment of the engineer must be relied upon to a considerable extent in drawing the distinctions between these different classes of ore. In general, however, developed ore is restricted to ore already broken in the stopes plus ore exposed on four sides—that is, ore that actually can be measured. Of course, if the exposed faces of a block of ore are far apart, an assumption is necessary regarding the continuity of ore between them, and the accuracy of the result again depends on the estimator's judgment.

In estimating probable ore, somewhat more latitude is allowed. It is usual, however, to class as probable ore only that exposed and measurable on at least two sides, or measurable on one side and exposed at one point on the opposite side of a block. In the latter instance it is customary to compute a tonnage based on the area of a triangle having as its base the exposed side and its apex the exposed point on the opposite side. In the probable ore tonnage, there also may be included some ore extending beyond exposed faces, provided known geological conditions warrant the expectation that the ore will continue a given conservative distance below a level or beyond a drift face.

In estimating possible ore, still more latitude may be permissible. The estimator will be influenced by past experience as to extensions of ore if the mine is an old one and by geological conditions known to affect localization of ore shoots.

Since the greatest present value of a mining property actually may lie in its potential ore (probable and possible ore) rather than in its developed ore, estimates of the former may be of utmost importance. This value can hardly be computed by formulas or by any fixed rules, and the judgment of the estimator, his experience, knowledge, and integrity comprise the only measure of the reliance that can be placed on his appraisal. In any event, since the estimated tonnages and grades of probable and possible ore are more subject to error than are those of fully developed ore, it is logical to set up ore reserves under three classes to indicate the degree of reliance that can be placed on each.

ASSAY MAPS AND SECTIONS

Assay plans may be prepared in various ways, depending on the types of the ore bodies. Whatever the method employed, the position of each sample, its length, and its assay value should be posted on the maps or sections. If the plans are to be submitted as part of a report that is to be or may be checked by another estimator, it is important that they be posted in this detail. Engineers often are called upon to check reports in which these details are omitted and only the results of computations of the original estimator are given. In some instances the figures merely show the computed tonnage and average grade of each of several blocks of ore. In others, the computed average assay values of each of the sides of the ore blocks or averages of groups of samples covering given lengths of ore exposures are shown, as, for example, "50 feet, 10 samples, average value $9.50, average width 6.5 feet." In such instances, the calculations cannot be checked in their entirety and the check estimator is left in the dark on such points as the treatment that has been accorded erratic high assays. As will be shown later, one erratic high assay averaged with several assays well below average ore grade may give an erroneous result considerably above the actual grade. Assay plans may consist of plan maps of the workings, drill holes, and ore bodies, vertical cross sections and longitudinal sections, or projections on the planes of the veins, or combinations of these.

VOLUME AND TONNAGE ESTIMATES

The plans are drawn to scale, and from them areas may be scaled or measured with a planimeter. Areas are commonly figured in square feet and ore volumes in cubic feet (except where the metric system is employed). Volumes in cubic feet are then converted into tons by dividing by the number of cubic feet in a ton of the rock or ore in place as determined by specific gravity or other tests.

VOLUME

For flat-lying, bedded deposits of uniform thickness, volume computations involve merely scaling or measuring with a planimeter the areas of the different blocks of ore bounded by mine workings and the margins of the ore bodies and multiplying the area of each block by the average thickness of the ore in that block. Summation of the volumes of all blocks gives the total volume of the ore.

For broad, basin-shaped deposits, plan maps, vertical or horizontal cross sections or both may be used. If the deposit has been explored by vertical drill holes or test pits, a common method is to plot the holes in plan and divide the area into a series of triangles, squares, or irregular polygons, each of which represents the horizontal cross section of an imaginary vertical prism of ore extending from the top to the bottom of the deposit. The volume of the prism is the area of the cross section multiplied by the depth, if the top and bottom of the prism are plane parallel surfaces. Since in deposits of this type the top and bottom may be very uneven, this method is an approximation; the closer the spacing of the holes, and thus the greater the number of prisms, the more closely will the approximation approach the actual volume. The sum of the volumes of all the prisms is then the total volume of ore in the deposit.

Sometimes holes are drilled at regular intervals at the apexes of equilateral triangles or at the corners of equal squares. More often, on account of topography or irregularities of the deposit, this is not possible and the arrangement is very irregular. The areas of influence or "weights" of the different holes then vary, sometimes considerably, with respect to both volume and ore-grade calculations. Complicated methods for constructing geometric figures to determine these factors have been described and discussed in textbooks and the technical press. However, competent mathematicians consulted by the authors agree that none of them is mathematically correct for all drill-hole arrangements and variations in grade. On the other hand, simple constructions are possible from which results may be obtained that are well within the order of accuracy of other factors entering into ore-reserve calculations; for example, errors in sampling and in assumptions as to ore boundaries may exceed the errors in approximations derived from these simpler methods.

Since the method of constructing these figures also affects the calculations of ore grade, this subject will be discussed further under the caption, Estimation of Grade of Ore.

The volume of large basin-shaped deposits may also be estimated from parallel cross sections on which are plotted intersections or projections of mine workings, drill holes, and ore boundaries. It is often convenient to draw vertical cross sections through lines of drill holes, raises, and winzes, for which purpose it may be necessary to prepare longitudinal sections and plans, or bottom- and top-of-ore contours from which points on the cross sections may be derived by interpolation. Having prepared a series of parallel cross sections, the area of each is computed or is measured by planimeter. If the variation in size and shape of adjoining sections is small, the volume in cubic feet of the block between them may be taken, for all practical purposes, as their average area multiplied by the distance between them in feet. If there is a series of such sections spaced equidistantly, the total volume may be expressed by the "end-area" formula:

$$V=\frac{1}{2}\,H(A_1+2A_2+2A_3+\;....\;A_n)$$

where A_1, A_2, A_3 ..... A_n are the areas of the several sections,
H is the perpendicular distance between them,
and V is the total volume.

If the ore body is very irregular and adjoining sections exhibit marked dissimilarity in size and shape, the volume between two adjoining sections may be approximated more closely by using the prismoidal formula:

$$V=\frac{H}{6}\,(A_1+4M+A_2),$$

where V=volume of the block,
H is the perpendicular distance between A_1 and A_2, A_1 and A_2 are the areas of the two sections,
and M is the area of a parallel section halfway between A_1 and A_2.

M is determined by constructing a midsection interpolated from corners of sections A_1 and A_2, modified by any information available concerning outlines between A_1 and A_2 that may be obtained from mine workings or geological data.

Figure 10 shows two adjacent sections that are quite dissimilar in shape. By using the end formula, the volume is 17,737,500 cubic feet, but by using the prismoidal formula it is 18,162,500, a difference of 425,000 cubic feet, or 2.5 percent.

If enough data are available to construct sections at close intervals, the prismoidal formula may be used, making each alternate section a midsection.

For dipping veins or lenses and other tabular forms of deposits, estimates are made from level plans, cross sections, and longitudinal projections on the plane of the dip, drawn to scale. All the assay widths and values are shown on these drawings, together with mine workings and outlines of blocks of ore remaining as determined by development or assumed from geological and other controlling data. The areas of the different blocks of ore multiplied by the corresponding average thickness of each, as determined from measurements normal to the dip, give the volumes of the respective blocks.

TONNAGE

Tonnage estimates are derived from those of volume by applying a specific gravity or "tonnage" factor. The specific gravity of the mineral mass may be calculated roughly from the specific gravity and percentage of each of the constituent minerals and the porosity of the ore, or by making direct specific-gravity tests on representative samples from various parts of the ore body. Cubic feet per ton is then obtained by dividing 2,000 (2,240, if long tons are desired) by the product of 62.5 (weight per cubic foot of water) multiplied by the specific gravity of the ore $\left(\frac{2{,}000}{62.5 \times \text{sp. gr.}}\right)$. A more accurate tonnage factor may be determined by actually weighing the ore from measured volumes of excavation and averaging the results. A precise factor cannot, as a rule, be obtained for most large ore bodies because of the variations in mineral composition and in porosity from point to point, but a factor accurate enough for all practical purposes usually can be derived by the methods described.

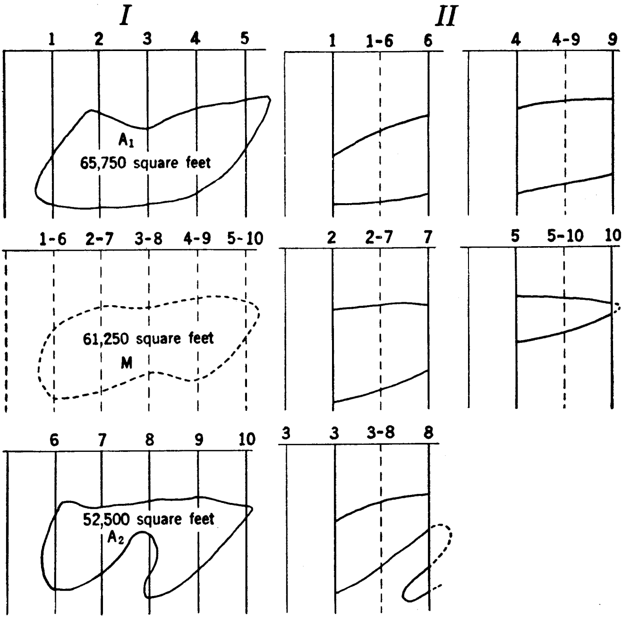

FIGURE 10.—Sections of irregular ore body used in computing volume of ore: 1, Cross sections. 2. Longitudinal cross sections for interpolation.

In some instances there may be enough variation in the specific gravity of different grades of ore to require different factors for each. Thus, in the Lake Superior iron-ore districts curves have been constructed showing the variation in cubic feet per ton with variations in iron content. One of these is shown in figure 11, *A*. At Cananea, Mexico, the specific gravity has been found to vary considerably with the copper content (fig. 11, *B*).

When these variations are substantial and are approximately known, it may be advisable, first, to calculate from the assay plans the average grade of each block of ore and then calculate the tonnage by employing the tonnage factor applicable to each grade. The tonnage in each block of ore is, of course, the volume in cubic feet divided by the cubic-feet-per-ton factor (or similar calculations for metric units).

When it comes to actual mining, it often happens that the tonnage is greater and the grade of ore lower than estimated. This often is due largely to dilution from the walls in stoping, though it may be due partly to mining of marginal ore of lower grade than that included in the estimates. There may be compensating factors, such as offshoots of high-grade ore not known to exist at the time of the estimate or

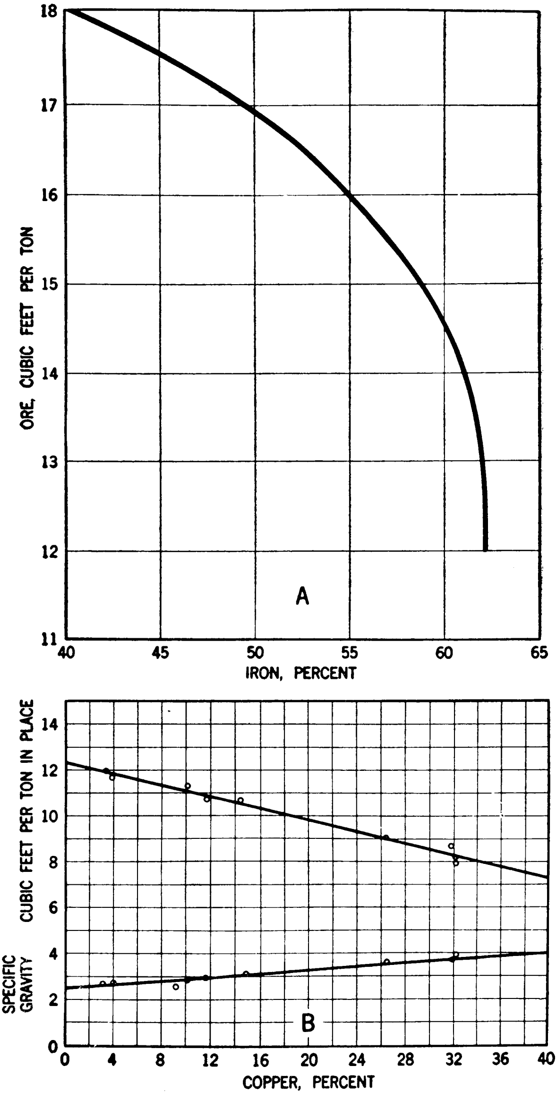

FIGURE 11.—Variation in cubic-feet-per-ton factor with grade of ore. *A*, Diagram for estimating tonnage factor for certain iron ores. *B*, La Colorada (Mexico) ore: Specific gravity and cubic feet per ton in place with varying percentage of copper.

unexpected inclusions of waste within the ore bodies. Allowance usually is made for possible or probable inclusion of waste and loss of ore in mining, by deducting a certain percentage from the calculated tonnage. Probable dilution from the walls may be compensated for by adding a certain percentage to the tonnage estimate and correspondingly reducing the calculated grade of ore. These percentages may be arbitrary or may be based upon experience with deposits of the particular type in question and with the mining method to be used. Thus, in one district experience has shown that a 10-percent deduction should be made for loss of ore in mining and another 10 percent for inclusion of waste. In another district it is necessary to allow for 20 percent dilution from the walls during mining; that is, the calculated tonnage must be divided by 0.80 and a corresponding reduction made in the calculated grade of ore.

COMPUTATION OF GRADE OF ORE

As previously indicated, the accuracy of ore-grade estimates will depend on the accuracy of the samples, accuracy in assaying, the method of combining and weighting the different samples, and the estimator's judgment.

It has been shown that individual samples may be relatively inaccurate, but the weighted average of a large number may closely approximate the actual grade of the ore. Accuracy in assaying may be achieved with care and correct laboratory technique. The proper combining and weighting of sample assays may require considerable thought, and correct mathematical principles should be employed, at least up to the point where the mathematical precision equals the accuracy of the data upon which the calculations are based.

ESTIMATES BASED ON DRILL-HOLE DATA

Sometimes it may be necessary, before combining and averaging the assay values of the different samples, to make some preliminary computations. This applies particularly where the samples have been obtained by core drills, if the core recovery is incomplete, or if there is an appreciable difference between core and sludge assays. In such instances it becomes necessary to average the assays of core and of sludge and in doing so to assign proper weight to the value of each. In weighting, certain assumptions must be made, and therefore the results will not be precise. If it is assumed that the hole and core are uniform in diameter over the length of the sample, that there have been no additions of material by caving in the upper portions of the hole, and that all of the cuttings (sludge) have been caught, simple arithmetical calculations can be made. In such case the cuttings will represent all that part of the material removed that is not core, and the weights assigned to core and sludge assays will be proportional to their respective masses. The relative masses have been calculated for standard sizes of diamond-drill bits and different lengths of core recovery from a 5-foot hole and tabulated. Table 10 has been prepared by the E. J. Longyear Co.[69] It is seen that even with complete core recovery the mass of the cuttings is greater than that of the core when the smaller bits are used.

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[69] E. J. Longyear Co., Minneapolis, Minn.: Catalog 8, 1931, p. 12.

TABLE 10.—*Percentages of volumes of core and cuttings in a 5-foot diamond-drill sample for each inch of core recovered, to be used as multipliers in combining analyses of core and cuttings*

Inches of core	EX[1]		AX[2]		BX[3]		NX[4]	
	Core	Cuttings	Core	Cuttings	Core	Cuttings	Core	Cuttings
1	0.6	99.4	0.6	99.4	0.8	99.2	0.9	99.1
2	1.2	98.8	1.2	98.8	1.6	98.4	1.7	98.3
3	1.8	98.2	1.8	98.2	2.4	97.6	2.6	97.4
4	2.4	97.6	2.4	97.6	3.2	96.8	3.4	96.6
5	3.0	97.0	3.0	97.0	4.0	96.0	4.3	95.7
6	3.5	96.5	3.6	96.4	4.8	95.2	5.1	94.9
7	4.1	95.9	4.2	95.8	5.6	94.4	6.0	94.0
8	4.7	95.3	4.8	95.2	6.4	93.6	6.8	93.2
9	5.3	94.7	5.4	95.6	7.2	92.8	7.7	92.3
10	5.9	94.1	6.0	94.0	8.0	92.0	8.5	91.5
11	6.5	93.5	6.6	93.4	8.8	91.2	9.4	90.6
12	7.1	92.9	7.2	92.8	9.6	90.4	10.2	89.8
13	7.7	92.3	7.8	92.2	10.4	89.6	11.1	88.9
14	8.3	91.7	8.4	91.6	11.2	88.8	11.9	88.1
15	8.9	91.1	9.0	91.0	12.0	88.0	12.8	87.2
16	9.5	90.5	9.6	90.4	12.8	87.2	13.7	86.3
17	10.1	89.9	10.2	89.8	13.6	86.4	14.5	85.5
18	10.6	89.4	10.8	89.2	14.4	85.6	15.4	84.6
19	11.2	88.8	11.4	88.6	15.2	84.8	16.2	83.8
20	11.8	88.2	12.0	88.0	16.0	84.0	17.1	82.9
21	12.4	87.6	12.6	87.4	16.8	83.2	17.9	82.1
22	13.0	87.0	13.2	86.8	17.6	82.4	18.8	81.2
23	13.6	86.4	13.8	86.2	18.4	81.6	19.6	80.4
24	14.2	85.8	14.4	85.6	19.2	80.8	20.5	79.5
25	14.8	85.2	15.0	85.0	20.0	80.0	21.3	78.7
26	15.4	84.6	15.6	84.4	20.8	79.2	22.2	77.8
27	16.0	84.0	16.2	83.8	21.6	78.4	23.1	76.9
28	16.6	83.4	16.8	83.2	22.4	77.6	23.9	76.1
29	17.1	82.9	17.4	82.6	23.2	76.8	24.8	75.2
30	17.7	82.3	18.0	82.0	24.0	76.0	25.6	74.4
31	18.3	81.7	18.6	81.4	24.8	75.2	26.5	73.5
32	18.9	81.1	19.2	80.8	25.6	74.4	27.3	72.7
33	19.5	80.5	19.8	80.2	26.4	73.6	28.2	71.8
34	20.1	79.9	20.4	79.6	27.2	72.8	29.0	71.0
35	20.7	79.3	21.0	79.0	28.0	72.0	29.9	70.1
36	21.3	78.7	21.6	78.4	28.8	71.2	30.7	69.3
37	21.9	78.1	22.2	77.8	29.6	70.4	31.6	68.4
38	22.5	77.5	22.8	77.2	30.4	69.6	32.4	67.6
39	23.1	76.9	23.4	76.6	31.2	68.8	33.3	66.7
40	23.7	76.3	24.0	76.0	32.0	68.0	34.2	65.8
41	24.2	75.8	24.6	75.4	32.8	67.2	35.0	65.0
42	24.8	75.2	25.2	74.8	33.6	66.4	35.9	64.1
43	25.4	74.6	25.8	74.2	34.4	65.6	36.7	63.3
44	26.0	74.0	26.4	73.6	35.2	64.8	37.6	62.4
45	26.6	73.4	27.0	73.0	36.0	64.0	38.4	61.6
46	27.2	72.8	27.6	72.4	36.8	63.2	39.3	60.7
47	27.8	72.2	28.2	71.8	37.6	62.4	40.1	59.9
48	28.4	71.6	28.8	71.2	38.4	61.6	41.0	59.0
49	29.0	71.0	29.4	70.6	39.3	60.7	41.8	58.2
50	29.6	70.4	30.0	70.0	40.1	59.9	42.7	57.3
51	30.2	69.8	30.6	69.4	40.9	59.1	43.5	56.5
52	30.7	69.3	31.2	68.8	41.7	58.3	44.4	55.6
53	31.3	68.7	31.8	68.2	42.5	57.5	45.3	54.7
54	31.9	68.1	32.4	67.6	43.3	56.7	46.1	53.9
55	32.5	67.5	33.0	67.0	44.1	55.9	47.0	53.0
56	33.1	66.9	33.6	66.4	44.9	55.1	47.8	52.2
57	33.7	66.3	34.2	65.8	45.7	54.3	48.7	51.3
58	34.3	65.7	34.8	65.2	46.5	53.5	49.5	50.5
59	34.9	65.1	35.4	64.6	47.3	52.7	50.4	49.6
60	35.5	64.5	36.0	64.0	48.1	51.9	51.2	48.8

[1] Approximate diameter of hole, 1½ inches; approximate diameter of core, ⅞ inch.
[2] Approximate diameter of hole, 1⅞ inches; approximate diameter of core, 1⅛ inches.
[3] Approximate diameter of hole, 2⅜ inches; approximate diameter of core, 1⅝ inches.
[4] Approximate diameter of hole, 3 inches; approximate diameter of core, 2⅛ inches.

When the core from a given section or "pull" is broken into a number of pieces it may not be possible to measure accurately the length of core recovered. As a check against the measurement, the core may be weighed, and this weight divided by the weight per inch of a core of the same diameter (calculated or determined from actual weighing) will give the equivalent length of core recovery.

The method of using the table may be illustrated by a hypothetical example. Assume a 5-foot section of EX diameter hole, 48 inches of core, a core assay of 3.0 percent copper, and a sludge assay of 3.6 percent copper. The core assay would receive a weight of 28.4 and that of the sludge 71.6. The calculation then becomes:

$$\text{Average assay}=\frac{1}{100}\,(28.4\times 3.00+71.6\times 3.6)=3.43 \text{ percent copper.}$$

If a portion of the cuttings is lost or other material is added from above a section of the hole under consideration, or if the hole is larger in diameter in softer portions of drill-hole section than in the harder sections, the weightings indicated by the table will not give correct average values when there is considerable difference between the assay-values of core and cuttings. Stated another way, and quoting from Longyear:

> If a portion of the cuttings is lost or other material is added, the sample will be correct only to the extent that the material analyzed represents a true sample of the actual cuttings.

In actual practice complete sludge recovery, entire absence of salting by caving, and holes of uniform diameter probably are the exception rather than the rule. These conditions, however, may be approximated in many instances so that no serious error is introduced by employing the proportional-volume method of combining sludge and core assays.

When there is considerable deviation from these ideal conditions, the estimator must use his judgment in combining core and sludge assays. At some mines it is the practice to disregard the sludge entirely, in others to apply a factor to the assays of core or sludge, or a combination of both, this factor being based upon experience already gained with the particular ore in question.

A serious error may be introduced in some cases if sludge assays are ignored. For example, consider an ore composed of alternate hard and soft layers, in which the soft layers contain a considerably higher percentage of valuable mineral than do the hard layers. In this instance, the core usually will be of smaller diameter and the outside diameter of the hole will be larger in the soft layers than in the harder layers. There will then be too small a porportion of the high-grade material in the core and too large a proportion in the sludge.

At Morenci, Ariz.,[70] the practice was to calculate the average copper content by combining the assays of core and sludge in proportion to their respective weights. If the weight of the sample did not correspond with the calculated weight of the corresponding section of drill hole, the results were not given full weight and adjustments were made in the calculated grades.

In Northern Rhodesia [71] a method of combining core and sludge assays based upon the following formula was developed:

Percent copper in core and sludge equals (weight of total core times percentage of copper in combined core plus weight of sludge times percentage of copper in sludge) divided by (weight of core plus weight of sludge).

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[70] Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Bureau of Mines Inf. Circ. 6107, 1929, p. 6.

[71] Matson, H. T., and Wallis, G. Allan, Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia: Trans. Am. Inst. Min. and Met. Eng., General Volume, 1931, pp. 66–82.

Having prepared the necessary plans and cross sections and placed thereon the assays of the drill-hole samples, which may have to be computed in some instances as outlined above, the next step is the combining of assays to obtain the average grade of different blocks or layers of ore and finally the average grade of the calculable ore reserves.

There are two general types of deposits of large area prospected by vertical drill holes: (1) Massive, irregular, basin-shaped, or flat-lying deposits with irregular or uniform mineralization, and (2) basin-shaped deposits in which different grades of ore are present in distinct layers.

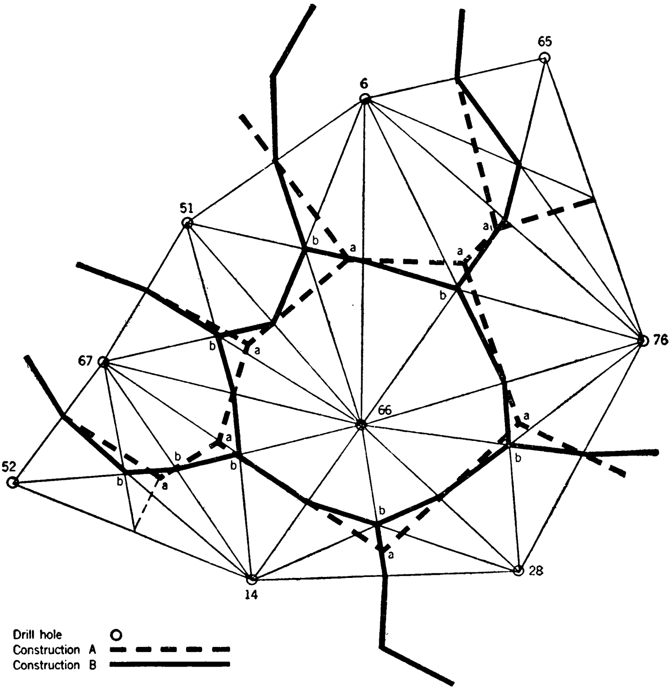

FIGURE 12.—Construction of polygons around drill holes for estimating tonnage and grade of ore.

CASE 1

In the first case, the triangular or polygonal prism method may be employed for computing tonnages and grades of ore. With this method each prism of ore is assumed to have the average grade of the drill hole around which it is constructed, and the geometrical construction should be such that this assumption will produce a result as nearly in accordance with the actual facts as possible.

There are two general methods of construction based on somewhat different hypotheses, viz: *A*. "That between any two sampling columns, the grade of ore of either one is assumed to extend halfway, and halfway only, to the other; and that in any prism, the grade of ore at any point is assumed to be that of the nearest assayed sample."[72] *B*. That variations in grade from one assayed sample point to the next are "straight-line" variations; that the grade at any point in a plane through three assayed sample points is influenced by the grade of all three in inverse ratio to the distance from each.

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[72] Harding, James E., How to Calculate Tonnage and Grade of an Ore Body: Eng. and Min. Jour.-Press, vol. 116, No. 11, Sept. 15, 1923, pp. 445–448.

The constructions based on these two hypotheses are also different and produce different results, as indicated by figure 12. With either construction, however, the procedure is mechanical and eliminates the personal equation in computation, and two different estimators using the same method will arrive at the same answer. Figure 12 shows in plan a number of drill holes forming part of a larger drill-hole pattern. The broken lines show the construction and resulting polygon around hole 66, based on hypothesis *A*, whereas the solid lines show the polygon resulting from construction *B*.

In both constructions, the first step is to draw lines joining each hole to every other contiguous hole to form a series of triangles. In construction *A*, perpendiculars are erected from the centers of each side of each triangle intersecting at a common point as "*a–a* . . . *a*." By joining these points, a polygon is formed around each hole, which is assumed to represent the cross section of a prism of ore of the same grade as the hole. It may be noted that with this construction, *a* may fall entirely outside its triangle in certain cases, as at a_1 (obtuse triangles).

In construction *B*, a line is drawn from each apex of a triangle to the center of the opposite side, the three lines intersecting at a common point, *b*, in the center of the triangle and dividing it into three smaller equal triangles. If the grade at the center of the large triangle, as determined by relative distances from each apex, be multiplied by the area of triangle, the result will be the same as that obtained by multuplying the grade of each hole by one-third the area of the large triangle and adding the products. The broken heavy and solid heavy lines show the respective polygons and the light lines the constructions for *A* and *B*. The average grade of each hole is the sum of the products of each sample length by its assay value divided by the sum of the lengths. This grade, multiplied by the volume of the prism and divided by the volume-per-ton factor, gives the tons-percent or ton-dollars in the prism, as the case may be. Adding these products for all the prisms and dividing by the sum of the tonnages gives the computed average grade of the deposit.

CASE 2

In the second case, vertical cross sections of the deposit are best adapted to making the estimates. The drill holes are drawn on the sections, the samples and assay values are plotted, and points in adjoining holes having similar analyses indicating portions of the same ore layer are joined by lines, thus marking the tops and bottoms of the layers (fig. 13).

For any section, the average assay value of a layer is computed as follows: The average assay value of each hole for the layer in question is assigned to the area extending from the hole halfway to the holes on each side; the sum of the products of each area times the corresponding assay value, divided by the sum of the areas, gives the average assay of the layer on that section. The average assay value and area of the corresponding layer are calculated for all the cross sections in the same manner. The volume of the layer in the block between two sections is found by the end-area formula or by the prismoidal for-

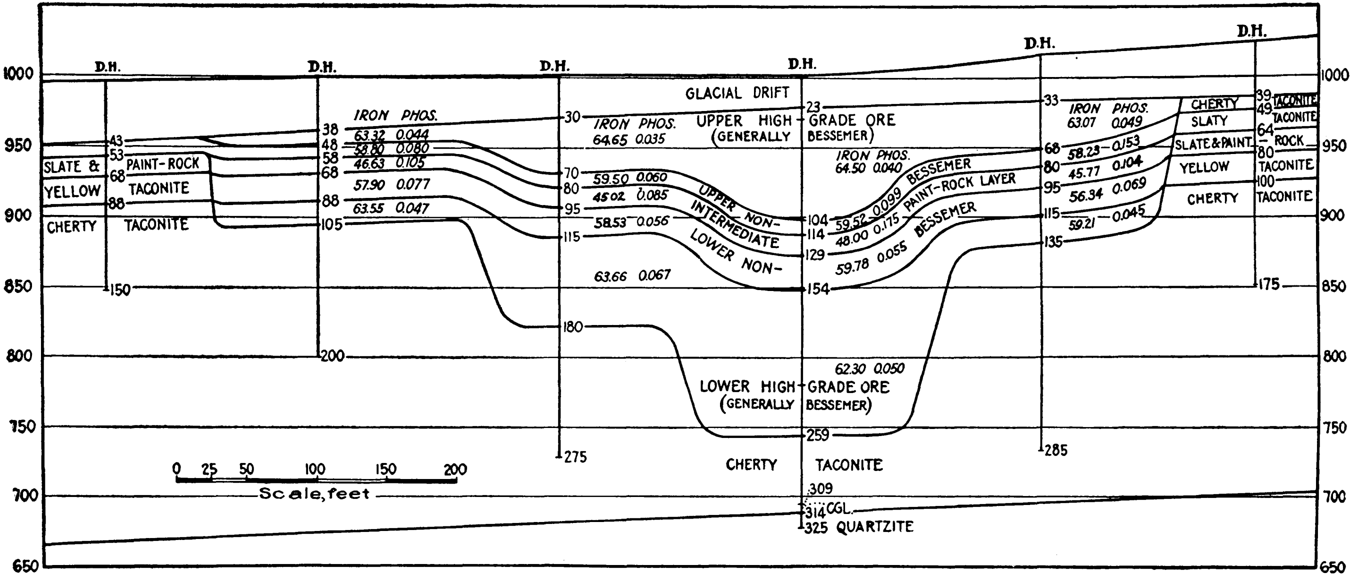
1000
950
900
850
800
750
700
650
D.H.
D.H.
D.H.
D.H.
D.H.
D.H.
GLACIAL DRIFT
UPPER HIGH
(GENERALLY
GRADE ORE
BESSEMER)
IRON PHOS.
63.32 0.044
58.80 0.080
46.63 0.105
57.90 0.077
63.55 0.047
IRON PHOS.
64.65 0.035
59.50 0.060
45.02 0.085
58.53 0.056
63.66 0.067
IRON PHOS.
64.50 0.040
59.52 0.099
48.00 0.175
59.78 0.055
IRON PHOS.
63.07 0.049
58.23 0.153
45.77 0.104
56.34 0.069
59.21 0.045
SLATE &
YELLOW
CHERTY
PAINT-ROCK
TACONITE
TACONITE
UPPER NON-
INTERMEDIATE
LOWER NON-
BESSEMER
PAINT-ROCK LAYER
BESSEMER
CHERTY
SLATY
SLATE & PAINT-ROCK
YELLOW
CHERTY
TACONITE
TACONITE
TACONITE
TACONITE
TACONITE
62.30 0.050
LOWER HIGH
(GENERALLY
GRADE ORE
BESSEMER)
CHERTY
TACONITE
CGL.
QUARTZITE
43
53
68
88
150
38
48
58
68
88
105
200
30
70
80
95
115
180
275
23
104
114
129
154
259
309
314
325
33
68
80
95
115
135
285
39
49
64
80
100
175
0 25 50 100 150 200
Scale, feet

FIGURE 13.—Cross section of a laminated-trough iron-ore body, showing typical structure and analysis of ore.

mula, and this divided by the volume-per-ton factor gives the corresponding tonnage. The average assay value of this block is computed then from the formula $V=\frac{(A_1V_1+A_2V_2)}{(\ \ A_1+A_2\ \ )}$; where V is the average value of the block, V_1 and V_2 are the values of the respective sections, and A_1 and A_2 are the corresponding section areas. Adding the products of the tonnages of the several blocks by their corresponding average assay values and dividing this sum by the sum of the tonnages gives the average assay value of the layer: $V=\frac{V_1T_1+V_2T_2+\ .\ .\ .\ .\ V_nT_n}{T_1+T_2+\ .\ .\ .\ .\ T_n}$, where V is the average value of the layer, V_1 . . . . V_n, the average value of the several blocks, and T_1 . . . . T_n, the corresponding tonnages in the blocks. Values and tonnages of the other layers of ore are computed similarly.

ESTIMATES BASED ON UNDERGROUND SAMPLING

It has been pointed out that in sampling mine workings, the samples should be taken at regular intervals to avoid errors that might result from introduction of the human factor in the selection of sampling points. With a regular interval the calculations of ore grade are simplified because no weighting of assay values for ore length is required.

WEIGHTING OF ASSAYS OF INDIVIDUAL SAMPLES

Each sample, however, must be weighted in proportion to the length of the sample cut across the ore body. Thus, referring to figure 7, if samples 1 to 5 are, respectively, 1.0, 2.5, 3.0, 2.75, and 2.8 feet long and the assay values are, respectively, \$0.50, \$5, \$20, \$4, and \$25, the average grade of the face or section is computed as follows:

First compute the average grade of the band represented by samples 3 and 5.

$$\begin{array}{lllll}
(3) & 3.0\times\$20 & = & \$60 & \$130\div5.8=\$22.41,\ \text{average grade.} \\
(5) & \underline{2.8}\times25 & = & \underline{\ \ 70} & 5.8\div2=2.9,\ \text{average width.} \\
 & 5.8 & & \$130 &
\end{array}$$

Then, for the entire section the calculation is:

$$\begin{array}{llll}
(1) & 1.0\times\$0.50= & 0.50 & \\
(2) & 2.5\times5.00 = & 12.50 & \\
(3\text{–}5) & 2.9\times22.41= & 65.00 & \\
(4) & \underline{2.75}\times4.00= & \underline{11.00} & \\
 & 9.15 & 89.00 & \$89.00\div9.15=\$9.72
\end{array}$$

Similarly, in averaging the assay values of a series of sample sections along the back of a drift, the sum of the products of the assay values of the several sections times the respective section widths divided by the sum of the section widths gives the average assay value of the series, provided the sections are equally spaced. If not equally spaced, the average may be computed by the formula

$$G=\frac{G_1L_1W_1+G_2L_2W_2+\ .\ .\ .\ .\ G_nL_nW_n}{L_1W_1+L_2W_2+\ .\ .\ .\ .\ L_nW_n}$$

where G is the average grade, G_1, G_2, etc., are the grades of the several sections, L_1, L_2, etc., the lengths of ore, and W_1, W_2, etc., the widths represented by the corresponding sections. G_1, G_2, etc., may be either assay values of single samples, where only one sample is taken for each section across the drift, or may be computed weighted averages where the drift section is composed of several samples representing different bands.

ERRATIC HIGH ASSAYS

When posting the assays on the assay plans and sections, occasional so-called erratic high assays may be found that require special treatment before they are entered in the computations. Although recurrence of such assays indicates the probable presence of high-grade spots or streaks in the ore body, which, if they occur often enough, may actually raise the average grade of the ore body appreciably, experience has proved that, if taken at their full value, the resulting estimates will be too high. When it is considered that in the majority of instances the erratic highs represent a condition occurring at only the particular spot where the sample was cut and are much higher than the average grade of the section or block of ore they influence, the need for caution in evaluating them can readily be understood.

Therefore, although mathematically there may be no sound reason for eliminating or reducing the high assays, the conservative practice is to reduce them in accordance with some empirical rule or by re-averaging. Often check-sampling of a high section may be necessary before recalculating.

Empirical rules may be based upon experience with the ore in a particular deposit. Thus, at the McIntyre mine in the Porcupine district of Ontario [73] erratic high assays among average low ones were arbitrarily cut to $50; where they are consistently high they are not reduced.

At the Hollinger mine, in the same district,[74] erratic high assays were cut to $100 in stopes known to be high-grade and to $50 in low-grade stopes. Where diamond-drill-hole assays are used in ore-reserve estimates, high assays are cut to $10. Ore showings in isolated diamond-drill holes are not included in ore-reserve estimates. Where the drill-hole sections are between faces of developed ore, the assays may be included, but high assays are cut to $10.

At these mines, the practices described have proved, over a period of years, to give accurate results.

Another practice employed at some mines is to average the erratic high assay with the assays of the two adjoining samples and insert this average in place of the erratic high in the final computation, as illustrated by the following example:

Sample number	Measuring point	Distance from measuring point, feet	Length of sample, inches	Assay value	Length times value
1001	Station 956	45 east	50	$8.00	$400.00
1002	do	50 east	30	[1] 50.00	1,500.00
1003	do	55 east	40	7.00	280.00
Total			120		2,180.00
Average [2]				18.17	

[1] Erratic high assay.

[2] $2,180.00 divided by 120.

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[73] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, p. 7.

[74] Jackson, Chas. F. and Knaebel, John B., Sampling and Estimation of Ore Deposits: Bureau of Mines Bull. 356, 1932, p. 126. Irving, Andrew Gordon, Mine Surveying [at Hollinger]: Canadian Min. Jour., September 1935, pp. 402-403.

The figure $18.17 is then inserted in place of $50 in the final computation.

A method virtually the same as the foregoing has been employed at the Wright-Hargreaves mine, Kirkland Lake, Ontario.[75] In estimating the grade of a given block of ore or averaging the assays along a section of a drift, all the assays, including the erratic highs at their full value, are first averaged by the dollar-foot method. A second average is then made, the first average being inserted in place of each erratic high. Values estimated in this manner have checked actual bullion recovery plus tailing loss within 5 percent; in some months the estimates are high and in others low, so that over a period of a year or more the average error is less than 5 percent (plus or minus). Following is a sample calculation in which round figures were used for grades and lengths of samples for the sake of simplicity.

First calculation, using actual assay values			Second calculation, substituting for erratic highs			First calculation, using actual assay values			Second calculation, substituting for erratic highs		
Feet	Assay value	Dollar-feet	Feet	Assay value	Dollar-feet	Feet	Assay value	Dollar-feet	Feet	Assay value	Dollar-feet
5	$6	30	5	$6	30	5	$8	40	5	$8	40
4	9	36	4	9	36	4	10	40	4	10	40
6	10	60	6	10	60	3	350	1,050	3	40	120
8	16	128	8	16	128	4	16	64	4	16	64
10	5	50	10	5	50	5	8	40	5	8	40
8	15	120	8	15	120	6	5	30	6	5	30
5	150	750	5	40	200	8	2	16	8	2	16
4	19	76	4	19	76	2	1,000	2,000	2	40	80
5	24	120	5	24	120	5	18	90	5	18	90
6	8	48	6	8	48	5	15	75	5	15	75
5	6	30	5	6	30	125	--------	5,000	125	--------	1,600
5	3	15	5	3	15						
4	8	32	4	8	32						
3	20	60	3	20	60	Average value $40			Average value $12.80		

COMPUTATION OF TONNAGE AND GRADE OF ORE BY BLOCKS

Developed ore usually is calculated first for each of the different areas or blocks bounded by drifts, raises, winzes, and outlines of stoped areas. The volume of each block is its area times the average thickness of the ore in the block. The volume divided by the volume-per-ton factor gives the tons of ore in the block. Multiplying the tonnage in each block by its average assay value, adding the products, and dividing this sum by the sum of the tonnages gives the average assay value of all the blocks.

In computing average thickness and average grade, it is important to weight each sample or series of samples in accordance with the length, area, or volume it represents.

Even where samples are spaced at regular intervals, serious error may result from incorrect combining of assays if the grades of the individual samples or sections vary greatly. This is shown by the example given in figure 14, where *a* and *b* are drifts 300 feet long between raises *c* and *d*, 100 feet long connecting *a* and *b*. The figures show the lengths and values in dollars of samples that are cut at equal intervals around the block. The first figure is the length of the sample (equal to the thickness or width of the ore in this case).

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[75] Jackson, Chas. F., and Knaebel, John B., work cited in footnote 74, pp. 128–129.

If, in this example, equal weight is given to the products (width times grade) for all the samples, including the raise samples, the tonnage (using a factor of 12 cubic feet per ton) will be 17,812, the average grade $7.98 per ton, and the total value of the ore in the block $142,140. The average grade of the raise samples along *c* is seen to be higher than that of the other samples, and if given equal weight to the drift samples by simple averaging, their weight would be out of proportion to their actual influence and result in a value that is too high.

If, however, the block is divided as shown by the lines *f f f f* and *g* and the resulting triangular and trapezoidal blocks are computed separately, the end or raise samples will then be weighted more in accordance with their actual influence and a slightly greater tonnage, a lower average grade, and smaller total value will be obtained. The calculation is as follows:

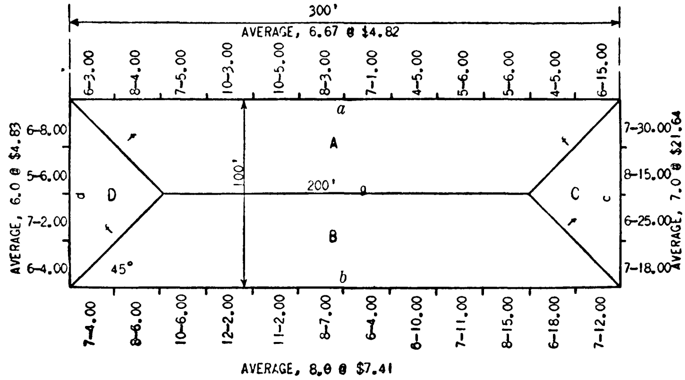
300'
AVERAGE, 6.67 @ $4.82
6–3.00
8–4.00
7–5.00
10–3.00
10–5.00
8–3.00
7–1.00
4–5.00
5–6.00
5–6.00
4–5.00
6–15.00
a
A
100'
200'
g
B
b
D
d
45°
C
c
f
f
f
f
AVERAGE, 6.0 @ $4.83
6–8.00
5–6.00
7–2.00
6–4.00
AVERAGE, 7.0 @ $21.64
7–30.00
8–15.00
6–25.00
7–18.00
7–4.00
8–6.00
10–6.00
12–2.00
11–2.00
8–7.00
6–4.00
6–10.00
7–11.00
8–15.00
6–18.00
7–12.00
AVERAGE, 8.0 @ $7.41

FIGURE 14.—Diagram showing division of block for estimating average grade of ore.

Block A.—$\frac{300+200}{2}\times\frac{50\times 6.67}{12}=6{,}948$ tons with an average grade of $4.82.

Block B.—$\frac{300+200}{2}\times\frac{50\times 8.0}{12}=8{,}333$ tons with an average grade of $7.41.

Block C.—$\frac{100\times 50}{2}\times\frac{7.00}{12}=1{,}458$ tons with an average grade of $21.64.

Block D.—$\frac{100\times 50}{2}\times\frac{6.00}{12}=1{,}250$ tons with an average grade of $4.83.

Tons		
6,948	@ $4.82	$33, 489
8,333	@ 7.41	61, 748
1,458	@ 21.64	31, 551
1,250	@ 4.83	6, 037
17,989	@ 7.38[1]	132, 825

[1] $132,825 ÷ 17,989.

REVISION OF COMPUTATIONS

In the foregoing discussions the computations are mathematical, and the results obtained generally have to be modified to allow for mining and other factors peculiar to the particular deposit under investigation.

Thus, where random, irregular horses of waste are known to occur within the ore bodies, a deduction may have to be made from the calculated tonnage. This may be a percentage deduction based on past experience in mining the particular deposit or other similar deposits in the district, or it may be an engineering safety factor to provide for possible waste inclusions.

The computed figures may indeed be quite accurate as to tonnage and grade of ore actually in the deposit but may be quite different from the tonnage and grade of ore that actually can be recovered by mining. Thus, support of the surface may require that a considerable percentage of the total ore be left permanently in the form of pillars. In structurally weak ore bodies, in particular, it is seldom possible to prevent the loss of some ore in mining. This loss will depend on the irregularity of the ore margins and other characteristics of the deposits and may vary with the particular mining method employed, which may depend, in turn, on economic considerations, safety in working, and other factors. It is clear, therefore, that computed tonnages usually must be reduced by amounts based on these considerations to determine the amount of recoverable ore in the deposit. Reductions to take care of ore losses may or may not affect the resulting average grade of the recoverable ore. When pillars must be left, it is often possible to leave them in leaner ore, thus resulting in a higher than average grade for the ore that can be extracted. Conservative engineers, however, seldom would raise the computed grade on this account.

On the other hand, dilution from the walls of the deposit in stoping may considerably increase the total tonnage that will be mined, but this results definitely in reduction of the recoverable grade below the computed grade in proportion to the amount of dilution. The amount of dilution will vary with the nature of the deposit and the enclosing wall rocks and with the stoping method employed.

Frequently, hand sorting underground or on the surface is resorted to for the purpose of raising the grade of the ore shipped or sent to the mill. Some loss of ore almost invariably results, and although the grade of the product is raised, the tonnage to be treated is reduced, and, other things being equal, the total amount of metal recovered may be less. Actually, the total recovery in some instances may be more, since sorting may enable profitable mining of considerable tonnages of ore of too low-grade to be included as ore in the ore-reserve estimates.

Another problem often met is the recalculation of tonnages and grades where the vein width is less than the practicable stoping width. In vertical veins having firm walls it is sometimes possible to mine widths as narrow as 2 feet with little overbreak. More often, however, 36 to 40 inches is the narrowest average width that can be mined. If the vein is narrower than the practicable stoping width there must obviously be an overbreak in mining, and values of samples cut only over the vein width must be reduced to equivalent values over stoping width. Thus, assuming a vein averaging 2 feet in width and $60 per ton and a minimum stoping width of 3 feet, the $60 grade spread over 3 feet then becomes two-thirds of $60, or $40. That is, the grade of the material broken will be $40, and if no sorting is done, the grade of ore shipped or sent to the mill will not exceed $40 per ton.

All such factors must be considered in making the final estimates of recoverable ore and later in comuting probable future profits and in appraising the value of the property, a subject that will be discussed in a later chapter.

MINE DEVELOPMENT

The term "mine development" is employed to designate the operations involved in preparing a mine for ore extraction. These operations include tunneling, sinking, crosscutting, drifting, and raising.

In most mines, both exploration and development continue after ore extraction by stoping has begun and often nearly to the end of the life of the mine. Although exploration and development operations are similar, the emphasis is placed on ore finding in the former, whereas in the latter operations are directed mainly toward preparation for economical extraction and removal of the ore from the mine.

As previously pointed out, it may often be possible to drive exploratory openings in such positions and in such a manner that they may be utilized later for ore extraction. In other instances, however, openings driven with a view to expeditious and economical exploration may not be well-adapted to meet economical development and ore extraction.

Thus, an exploratory shaft may be too small for hoisting ore, men, materials, and supplies during the productive period and to provide room for pipes, electric-power cables, and ladderways. Exploratory drifts and crosscuts may be too small in cross section or too crooked for economical motor haulage, or they may be situated improperly with respect to stope centers. In other instances they may be in a lode that is structurally weak and in which, therefore, their maintenance cost would be so excessive that permanent haulageways must be driven in one or the other of the walls.

In mines situated in rugged mountainous districts and explored through shaft openings it may sometimes be desirable, for operating purposes, to drive a long adit to replace the shaft as the main extraction opening to eliminate high pumping and hoisting charges or to provide the most economical means of transporting the ore to an ore-dressing plant. Sometimes it is necessary or desirable to replace an exploratory inclined shaft by a vertical-operating shaft or to substitute one in the footwall of the lode for one in the lode itself.

Among the important factors to be considered in planning mine development for economical operation are the following:

1. Location of the main extraction opening.
2. Shaft vs. adit or tunnel development and inclined vs. vertical shafts.
3. Time required to prepare for production and anticipated maximum rate of production.
4. Level interval.
5. Level-development plans and their adaptation to the physical characteristics of the deposits and to the stoping method to be employed.
6. Life expectancy of the mine, total tonnage to be mined, and comparative costs of different development schemes.

Obviously, the type of ore deposit, its size, shape, and dip, distribution of ore shoots or separate ore bodies, strength of ore and wall rocks, and surface topography are conditions imposed by nature. It becomes necessary, therefore, to fit the development plan to these conditions, taking the utmost advantage of those that are favorable. It is apparent that the more complete the information obtained from previous exploration, the more intelligently can development be planned.

LOCATION OF MAIN EXTRACTION OPENING

The main opening is usually a permanent one that will be utilized throughout the life of the mine. It should, therefore, be so situated relative to the ore bodies that it will not be disturbed by ground movement caused by ore extraction and with a view to low maintenance cost.

It is advantageous, other things being equal, to locate the main opening centrally with regard to the ore bodies so that underground transportation may be as economical as possible. A central location is not always possible because of the surface topography or the rake or pitch of the ore body. Thus, the topography may be such that construction of a surface plant near the main opening would not be feasible in a central location or that transportation of ore to the mill would be unduly expensive. The deepest ore may be at one end of the deposit, and in some instances it may outcrop at one end and plunge toward the other end, and the shaft obviously should be located nearer the deepest ore to avoid having to drive long haulageways in waste rock and sink auxiliary underground winzes or inclines.

The adequacy of existing exploratory openings for ore extraction should be considered, and naturally these should be utilized when it is practicable to do so.

The best position for the main opening should then be determined from a consideration of the topography, the position of the mill site or point of loading ore for shipment, the feasibility of constructing a surface plant consisting of hoist house, shops, change house, and other buildings near the opening, the "center of gravity" of the known ore, both in plan and vertically, the position of the deepest ore, and the position and adequacy of existing openings.

SHAFTS VS. ADITS

In areas of low relief, where most of the ore lies below surface-drainage levels, it is apparent that the deposits must be developed by shafts. In areas of high relief, where much of the deposit lies above surface drainage, there is often a choice between shaft and adit development, and the option selected may be influenced by a number of considerations. The surface topography may preclude sinking a shaft near the deposit, or the only mill site available may be so situated that an adit is indicated immediately.

The cost per foot of shaft sinking may be several times that for driving an adit, the ratio in any given case depending on the cross-sectional area of the shaft or adit, the strength and nature of the ground, the amount of water to be handled, and the cost of power for hoisting and pumping. The plant-installation cost for shaft sinking is usually considerably greater than for a tunnel, since hoisting equipment, headframe, and pumping equipment must be provided. This greater investment for equipment may be several times the extra cost of track and cars required for adit development.

A very important consideration is the amount of water that will have to be handled. This becomes increasingly important in a mine with a long life expectancy, because pumping charges in a shaft mine go on year after year. The power required for pumping increases directly with the volume of water and height of lift.

A tunnel or adit drains the workings above it by gravity flow; and even though the workings may later be extended below the adit level by means of an inside shaft, it frequently happens that most of the water enters the mine at the upper levels and the lower levels are quite dry. In any event, the height to which water from lower levels must be lifted is reduced. The expense of hoisting ore and waste, caging cars or loading skips, and handling men and supplies in shafts comprise an appreciable operating-cost item.

On the other hand, adit development may involve driving headings, the excessive length of which may more than offset the additional costs for shaft sinking and shaft operation.

With shaft development, it is almost universally the practice to sink, develop, and start stoping in stages of one or a few levels at a time as new ore faces are required to maintain production and as ore continuity at greater depth is proved or indicated.

With adit development, a similar plan may be followed by driving a number of successively longer adits at successively lower elevations. When these can be driven on the vein instead of across the strike, development by this method has obvious advantages. When they must be driven in waste, the time involved in reaching the ore, the cost of so much dead work, and the uncertainties regarding persistence of the ore bodies at horizons corresponding to the elevations of the adits may minimize or eliminate entirely the advantages of adit development.

These and other factors that may be peculiar to a given deposit must be weighed against each other.

INCLINED VS. VERTICAL SHAFTS

As far as shaft operations *per se* are concerned, vertical shafts usually are preferred to inclined shafts.[76] Inclined shafts must be longer to reach the same depth, hoisting speed is ordinarily slower, wear on hoisting ropes is greater, and there are usually rope idlers to install and keep in repair. In bad ground, vertical shafts usually are easier to maintain.

An inclined shaft may be more economical where first cost of development and speed with which the mine can be brought into production more than offset the operating economies effected by vertical shaft development. An incline in the foot or hanging wall (usually the former) can be driven approximately parallel to the plane of the lode, so that only short crosscuts are required on each level to reach the ore body. The flatter the dip of the lode, the greater is the advantage of an inclined shaft. It is obvious that a vertical shaft started in the footwall of a dipping deposit is farther from the lode on each successively lower level, so that the time required for crosscutting to the lode and the cost thereof may be prohibitive on the lower levels. If a vertical shaft is started in the hanging wall, it will be nearer the lode on successively lower levels, until it finally crosses it; the crosscuts will then increase in length as the depth increases. A shaft in the hanging wall may be affected by subsidence, resulting in high operating and maintenance costs; or, if large pillars of ore are left unmined below the shaft for its protection, such ore may be tied up so that it cannot be recovered until the mine is almost worked out.

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[76] Jackson, C. F., Some Notes on Underground Transportation in Metal Mines: Bureau of Mines Inf. Circ. 6326, 1930, pp. 23–24.

SIZES OF MAIN OPENINGS

The size (cross-sectional dimensions) of a shaft or main haulage adit will be governed to a large extent by the anticipated rate of production. Minimum size limits are established by practical considerations even for very small production rates. Thus an adit should be large enough to pass the mine cars and leave room on the side for a ditch and for a man to pass a car. If animals are employed for haulage, the adit must be high enough for them to pass; and if electric trolley locomotives are to be used, the adit should be high enough to permit suspension of the trolley wires well above the head of a man standing upright. If the production rate is to be high, large cars and locomotives usually are most economical for haulage, and the adit must be big enough to accommodate them; in long adits it may be advisable to double-track the haulageway or at least to widen it in places to make room for passing tracks.

A shaft should be large enough to provide at least one compartment for hoisting ore and handling men, materials, and supplies and a separate compartment for a ladderway, air and water pipes, and electric cables.

In the Tri-State zinc and lead district one-compartment vertical shafts 5 by 7 or 6 by 6 feet in cross section are common and ore is hoisted in "cans" or buckets holding about 1,200 to 1,400 pounds of ore. Here, however, the ore lies at comparatively shallow depths and there are usually one or more "field" shafts in addition to the hoisting shaft, which are available for additional exits from the mine, for ventilation, and sometimes for the handling of men, materials, and supplies. Some of the newer shafts in this district employ cages or skips for hoisting and are 5 or 6 by 11 or 12 feet in cross section, or larger.

In the Southeast Missouri lead district vertical shafts are usually about 6 by 15 to 6 by 18 feet in the clear (inside timber) and are divided into three compartments, two of which are used for hoisting with skips in balance and the other for pipes and power cables. Generally, there are auxiliary shafts for ventilation and for handling men and supplies.

In the Western States the usual shaft for small or medium production, where ore is to be hoisted in balance in cars on cages or in skips to obtain the power economy of balanced hoisting, is rectangular and a minimum of 5 by 15 feet inside the timber or 7 by 17 feet outside the timber. This provides for two hoisting compartments each 5 by 5 feet in the clear and a ladderway pipeway 3 by 5 feet in the clear.

Since the main operating shaft is often the main intake airway it should be of cross-sectional area great enough to pass at moderate velocities the amount of air required to ventilate the workings adequately. This is especially important in hot, deep mines. Sometimes auxiliary shafts are provided for intake or exhaust airways.

The first consideration in determining the size of a shaft is the duty to be required of it. This includes the hoisting of ore and waste and the hoisting and lowering of miners, timber, rails, explosives, drill steel and drilling equipment, and other materials and supplies. If the ore hoist also handles men and materials, the time available for hoisting ore obviously is reduced. The time required for these other duties depends on a number of factors. Thus, some mines use large amounts of timber for ground support, whereas others require very little; some use proportionately much greater quantities of drill steel than others; some shafts have to handle large quantities of waste for stope filling while others have little or no waste to handle. In some mines, especially those operating on a large number of levels, service demands for transferring bosses from one level to another and handling equipment consume a large amount of time. In such instances it is usually economical to make the shaft large enough to accommodate a separate man and service cage, unless the production is so small that a single hoist can handle it easily and perform service duties as well. Where large amounts of mine timber must be handled through the shaft, it may be desirable to provide a separate long cage upon which loaded timber trucks can be run with the timbers lying flat. The long cage is convenient, also, for handling drill-steel trucks with the steel lying flat.

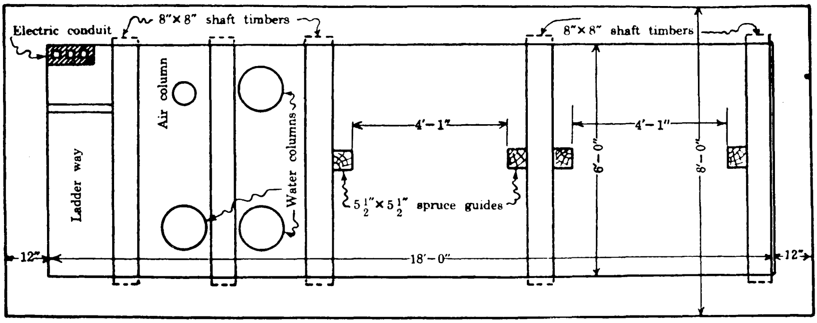

Electric conduit
8"×8" shaft timbers
8"×8" shaft timbers
Air column
Water columns
Ladder way
4'-1"
4'-1"
6'-0"
8'-0"
5 1/2"×5 1/2" spruce guides
12"
18'-0"
12"

FIGURE 15.—Plan of shaft, No. 8 mine, Southeast Missouri.

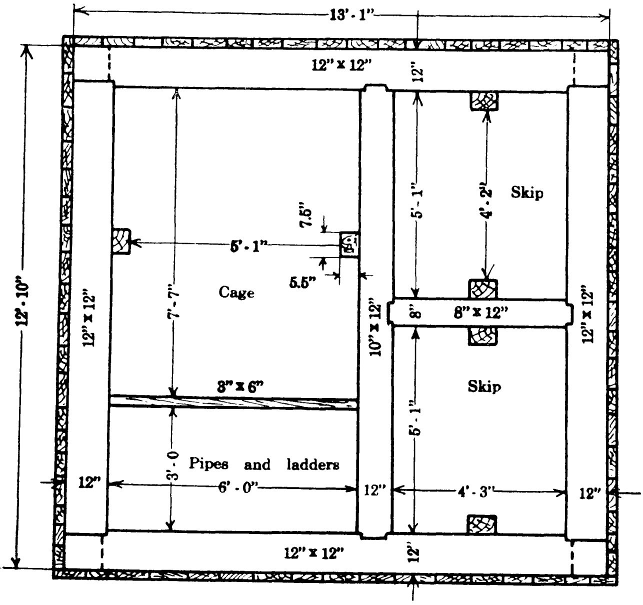

13'-1"
12"×12"
12"
12'-10"
12"×12"
7'-7"
7.5"
5'-1"
5.5"
Cage
8"×6"
3'-0
Pipes and ladders
12"
6'-0"
12"
10"×12"
5'-1"
4'-2"
Skip
8"
8"×12"
Skip
5'-1"
4'-3"
12"
12"×12"
12"×12"
12"

FIGURE 16.—Plan of shaft, Menominee No. 1 mine.

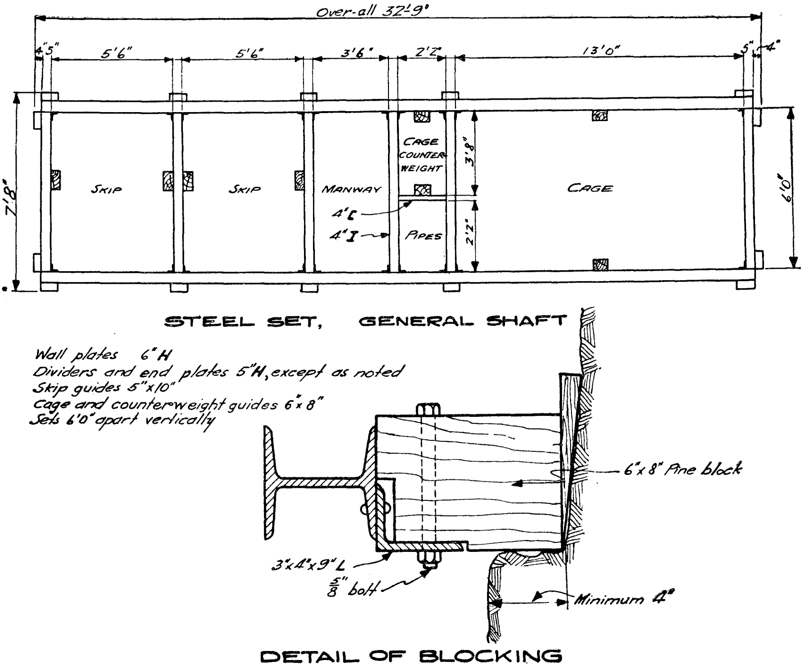

Over-all 32'-9"
4"
5"
5'6"
5'6"
3'6"
2'2"
13'0"
5"
4"
7'8"
6'0"
3'8"
2'2"
Skip
Skip
Manway
Cage counter weight
Pipes
Cage
4"⊏
4"I
STEEL SET, GENERAL SHAFT
Wall plates 6" H
Dividers and end plates 5" H, except as noted
Skip guides 5" x 10"
Cage and counterweight guides 6" x 8"
Sets 6'0" apart vertically
6" x 8" Pine block
3" x 4" x 9" L
5/8" bolt
Minimum 4"
DETAIL OF BLOCKING

FIGURE 17.—Plan of shaft, Fresnillo, Mexico.

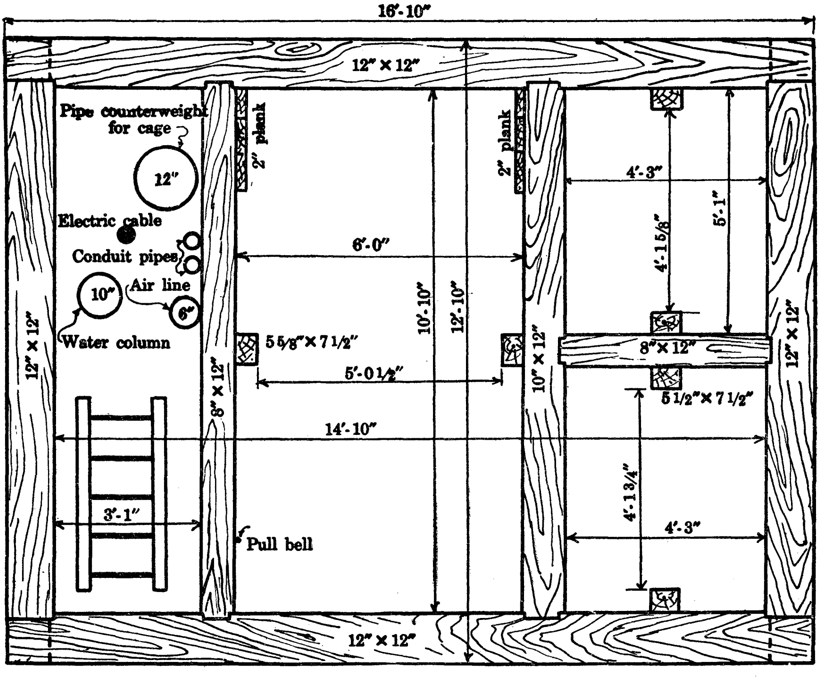

16'-10"
12" × 12"
Pipe counterweight for cage
12"
Electric cable
Conduit pipes
Air line
10"
6"
Water column
2" plank
2" plank
6'-0"
10'-10"
12'-10"
5 5/8" × 7 1/2"
5'-0 1/2"
14'-10"
3'-1"
Pull bell
4'-3"
5'-1"
4'-1 5/8"
8" × 12"
5 1/2" × 7 1/2"
4'-1 3/4"
4'-3"
12" × 12"
8" × 12"
10" × 12"
12" × 12"
12" × 12"

FIGURE 18.—Plan of shaft, mine No. 2, Marquette range.

The capacity of a shaft of a given size is usually greater when skips are employed for ore and waste than when cages are used; the net load and the ratio of net load to total load is greater, and less time usually is required for loading and dumping. Even when the ore hoist is not used for service work, it is seldom that hoisting can be conducted full time, especially if pockets of considerable capacity

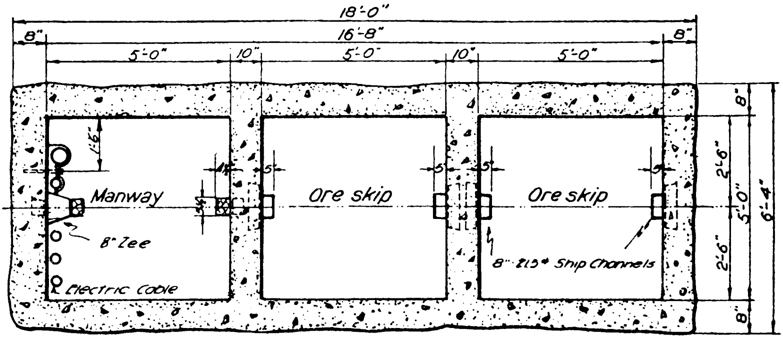
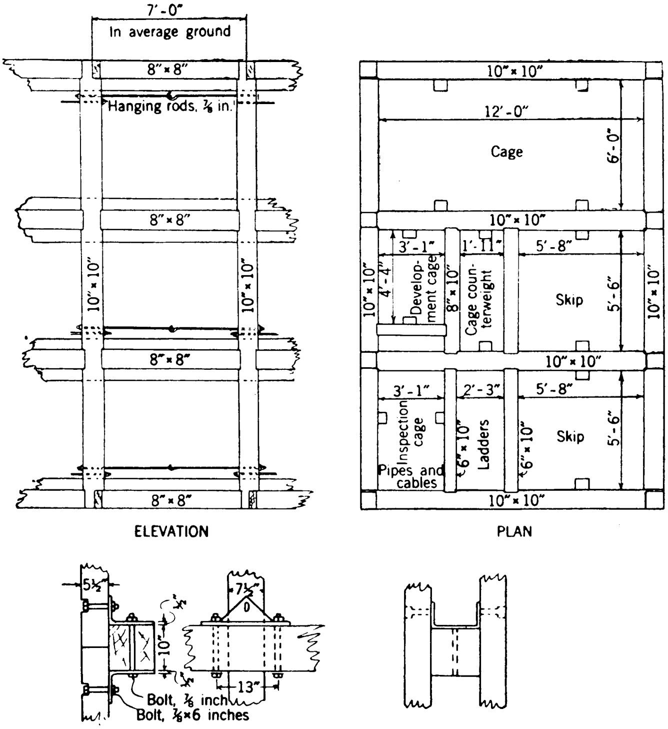

18'-0"
8"
16'-8"
8"
5'-0"
10"
5'-0"
10"
5'-0"
1'-6"
4½
5
3½
Manway
8" Zee
Electric Cable
Ore skip
Ore skip
8" 21.5# Ship Channels
8"
2'-6"
5'-0"
6'-4"
2'-6"
8"

FIGURE 19.—Plan of shaft, United Verde mine.


7'-0"
In average ground
8"×8"
Hanging rods, ⅞ in.
8"×8"
10"×10"
10"×10"
8"×8"
8"×8"
ELEVATION
10"×10"
12'-0"
Cage
6'-0"
10"×10"
3'-1"
1'-11"
5'-8"
4'-4"
Develop-ment cage
8"×10"
Cage coun-terweight
Skip
5'-6"
10"×10"
10"×10"
3'-1"
2'-3"
5'-8"
Inspection cage
6"×10"
Ladders
6"×10"
Skip
5'-6"
Pipes and cables
10"×10"
PLAN
5½
7½
10"
13"
Bolt, ⅞ inch
Bolt, ⅞×6 inches

NOTE: One bracket per guide on each set; two brackets at guide joints

FIGURE 20.—Timbering and arrangement of No. 11 shaft, McIntyre mine.

are not provided at each hoisting level. The provision of storage pockets of ample capacity, with measuring hoppers for loading the skips, materially promotes maximum utilization of time and shortens the hoisting cycle, hence increases the capacity of the shaft.

Hoisting in cars on cages has advantages under certain limited conditions, notably in leasing operations where the production from each of a number of sets of leasers must be kept separate from that of the others.

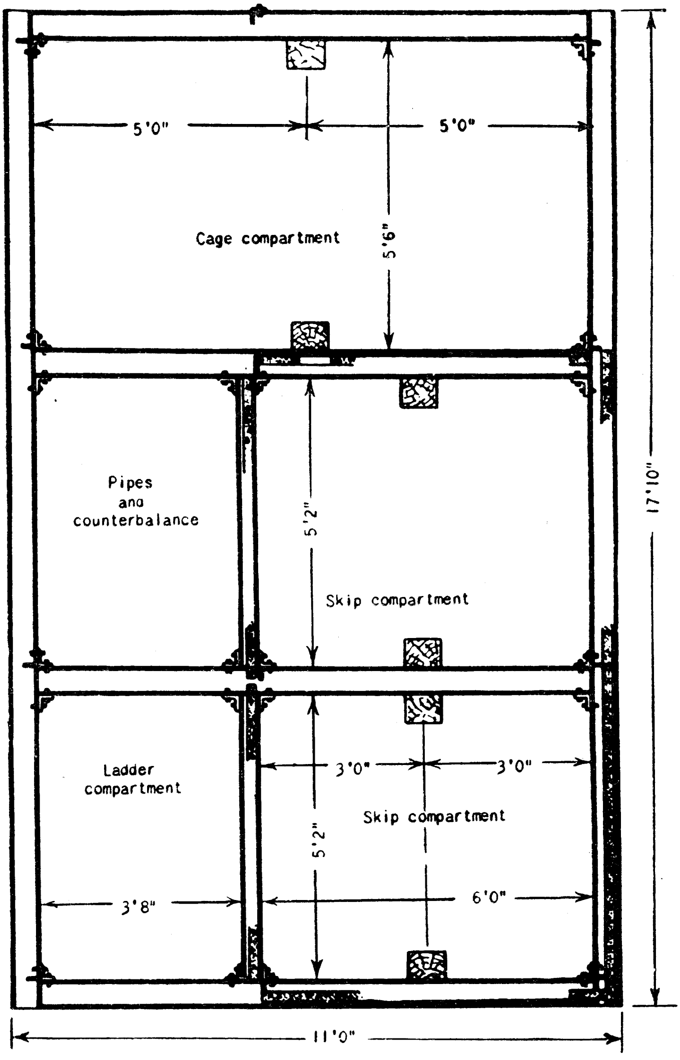

5'0"
5'0"
Cage compartment
5'6"
Pipes
and
counterbalance
5'2"
Skip compartment
17'10"
Ladder
compartment
3'0"
3'0"
Skip compartment
5'2"
3'8"
6'0"
11'0"

FIGURE 21.—Steel-timbered shaft, Montreal mine.

Tables 11 and 12 contain data on typical shafts, their size, number of compartments, daily capacity, and service duties.

TABLE 11.—*Data on typical vertical shafts*

Mine	Over-all dimensions in clear	Compartments	Ore capacity of skip or cage, tons	Approximate daily ore tonnage	Service duties	Remarks
Burra-Burra,[1] Tennessee, 74° incline.	6 ft. by 18 ft. 6 in.	Three; 2 hoisting	4-ton skips	1,600 ± ore	Separate shaft for men and supplies (1929).	Hoisting from 16 levels down to 1,600 feet.
Burra-Burra, Tennessee, vertical.	6 by 16 ft	do	do	Service only (1929)		
Barr,[2] Kansas	6 by 10 ft	Two for hoisting	Self-dumping cages, 1.5 tons.	450 to 500 ore	Separate auxiliary shafts	One level at 300 feet.
No. 8, Southeast[3] Missouri.	5 by 18 ft	Two for hoisting (see fig. 15).	4-ton skips hung under man cage.	550 to 600 ore	Slope for travel of men	586 feet deep.
Menominee No. 1,[4] Michigan.	10 ft. 8 in. by 11 ft. 3 in.	Two skips, one cage, one for pipes and ladders (see fig. 16).	4-ton skips	600 ore	Cage compartment	From levels at 200, 300, 400, 950, and 1,200 feet deep, respectively.
Hartley-Grantham,[5] Kansas.	6 by 6 ft	One for hoisting ore	Hoisting cans, 0.6 ton	400+	Separate shaft	One level at 200 feet.
Fresnillo,[6] Mexico	6 ft by 31 ft. 3 in.	Five; 2 ore skips, 1 long cage (fig. 17).	5-ton skips	Capacity 1,700 tons in 8 hours.		800 feet deep.
Mine No. 2, Marquette range,[7] Michigan.	10 ft. 10 in. by 14 ft. 10 in.	Two skips; one long cage; one pipe, cable, and ladders (fig. 18).	4-ton skips	Two shafts (total of 4 skips), 1,200 tons daily from 8 levels.	Separate shaft for ventilation and service.	8 levels; one shaft to 1,650 level, one to 800 level.
United Verde,[8] Arizona	5 ft. by 16 ft. 8 in.	Two skips, one manway (fig. 19).	2 of 112 cu. ft. each (8 tons sulfide ore, 6 tons schist ore).	2,000	Separate shafts	Hoist on 1,000-foot level. Hoists from all levels down to 3,000, level interval 150 feet and pulls ore to 800 level.
McIntyre,[9] Ontario, No. 11 shaft.	12 ft. by 18 ft. 8 in.	Two skips, 1 inspection cage, 1 development cage, 1 long cage, 1 ladderway, 1 cage counterweight (fig. 20).	6 tons	2,200	Long cage for men and materials; development cage for man service during shift; several old shafts for auxiliary purposes.	Hoists ore from 2,875 and 3,875 levels.
Hecla,[10] Idaho	5 ft. 2 in. by 17 ft. 4 in.	Two skips, 1 chippy cage, 1 pipeway.	7 tons	1,000 ore	Chippy cage, auxiliary raise, 1,600 to 2,000 level.	From 6 levels down to 2,000 feet (1929). Two classes ore and waste.
Black Rock (Butte and Superior),[11] Montana.	5 by 15 ft. plus chippy compartment 2,800 to 3,700 level.	Two skips; 1 pipes, cable, and manway.	do	1,200 capacity	Two auxiliary shafts, one for ventilation and one for supplies. Ore shaft has chippy cage 2,800 to 3,700-foot level.	
Inspiration,[12] Arizona	2 shafts each 5 ft. 6 in. by 19 ft. 2 in.	Two skips, 1 large man cage. Two skips, 1 compartment for pipes and ladders.	12.5 average	18,000 (three shafts)	Cages. Separate ventilation shafts.	Two shafts hoist from 585-foot level; one from 1,400-foot level.
	1 shaft 12 ft. 4 in. by 16 ft. 8 in.	Two skips; 1 large cage; 1 pipe and counterweight.				

Montreal,[13] Wisconsin	10 ft. by 16 ft. 10 in	Two skips; 1 long cage; 1 pipes and counterbalance; 1 ladders (fig. 21).	8 (long tons)	3,500 (ore only)	Two other shafts, inclined but same cross section.	Hoists four grades of ore from several levels, lowest 3,000 feet (1930).

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[1] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[2] Keener, O. W., Method and Cost of Mining at Barr Mine, Tri-State Lead and Zinc District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

[3] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

[4] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.: Bureau of Mines Inf. Circ. 6180, 1929, 10 pp.

[5] Keener, O. W., Methods and Costs of Mining at the Hartley-Grantham Mine, Tri-State Lead and Zinc District: Bureau of Mines Inf. Circ. 6286, 1930, 8 pp.

[6] Livingston, A., Mining Methods and Costs at Fresnillo, Zacatecas, Mexico: Bureau of Mines Inf. Circ. 6661, 1932, 31 pp.

[7] Eaton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6179, 1929, 15 pp.

[8] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[9] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, 18 pp.

[10] Foreman, C. H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Bureau of Mines Inf. Circ. 6232, 1931, 31 pp.

[11] McGilvra, D. B., and Healy, A. J., Methods and Cost of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Bureau of Mines Inf. Circ. 6370, 1930, 16 pp.

[12] Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

[13] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

TABLE 12.—*Data on typical inclined shafts*

Mine	Over-all dimensions in clear	Inclination	Compartments	Ore capacity of skip	Approximate daily ore tonnage	Service duties	Remarks
Burra-Burra, Tenn. (1928).[1]	6 ft. by 18 ft. 6 in	74°30′	2 skips; 1 ladder, pipes, and cables.	4-ton skip	1,500 to 1,600	Separate shaft partly.	Hoists from 13 levels, lowest 1,300. Pockets at 3 levels.
Nevada-Massachusetts, Nev. (2 shafts).[2]	4 ft. by 8 ft. 6 in	70° average	Each shaft has 1 skipway and a pipe and ladderway.	1-ton skip	250	All service through the two shafts.	Maximum depth, 1930, 730 feet; 1936, 1,150 feet.
Cortez, Nev.[3]	6 ft. by 12 ft	73°	Untimbered	1-ton skip	150±		Levels at 98 and 217 feet.
Mt. Hope, N. J.[4]	6 ft. by 19 ft. 4 in	64°	2 skips, 1 manway	48-cubic-foot skip.	500	All service	Hoisting from 1,000-foot level (1932).
Argonaut, Calif.[5]	5 ft. 9 in. by 13 ft. 11 in	70°	2 skips, 1 manway	4-ton skip	275	All service	Bottom level, 5,800 ft. incline distance.
Page, Idaho [6]	5 ft. by 16 ft. (approx.)	50°	2 skips, 1 pipe and manway.	2.3-ton skip	300	All service	Levels 300 feet apart down to 1,200 slope distance (1930).

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[1] McNaughton, C. H., Mining Methods and Costs of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[2] Heizer, O. F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Mines at Mill City, Nev.: Bureau of Mines Inf. Circ. 6284, 1930, 11 pp.

[3] Hezzelwood, G. W., Mining Methods and Costs at the Consolidated Cortez Silver Mines, Cortez, Nev.: Bureau of Mines Inf. Circ. 6327, 1930, 15 pp.

[4] Sweet, J. R., Mining Methods and Costs at the Mount Hope Mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp.

[5] Vanderburg, W. O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6311, 1930, 14 pp.

[6] Berg, J. E., Mining Methods at the Page Mine of the Federal Mining & Smelting Co., Page, Idaho: Bureau of Mines Inf. Circ. 6372, 1930, 8 pp.

METHODS, EQUIPMENT, AND COSTS OF DRIVING ADITS

Although practices in vogue for many years are still employed in small-scale operations, decided advances have been made, particularly in the last decade and especially in connection with large-scale operations. Contributing to this are introduction of mechanical loading equipment, mechanical ventilation, availability of cheap power, improved rock-drilling equipment, use of drill carriages and tenders in large headings, and mechanical haulage, all of which help to increase speed. The use of mechanical loading equipment permits pulling longer rounds at the same or lower total cost per foot.

Mechanization has not been brought about solely as the result of a desire to lower costs or to increase speed but, to an extent greater than is sometimes appreciated, by changing labor conditions. The younger generation of miners is more averse to arduous handwork than the older ones but takes kindly to operating machines.

EQUIPMENT

Equipment for driving is of four general types—drilling, blasting, loading and transportation, and ventilating.

Modern drilling equipment comprises rock drills; rock-drill accessories, including drill mountings, hose, and fittings; drill-steel and drill-sharpening equipment; air-compressor and motive power; and pipe lines for air and water. On some large jobs, mobile drill carriages and tenders also are included. Among the more recent innovations that are increasing in favor are detachable drill bits. When these are employed, a bit-grinder also may be required. The type and size of rock drill employed will depend on the nature of the ground. Where the rock is hard, 3-, 3½-, or 4-inch Leyner-type drifters are customary. In the softer ground, mounted jackhammer-type drills of about 2⅝-inch bore may be used. Quite recently automatic-feed drifters have been introduced and are finding favor on some jobs. Wet-drilling is or should be used in all adit work.

In large or moderately large headings and for use in fast driving, drill carriages have often proved advantageous. On these carriages are mounted the drill columns and arms, air and water manifolds with large hose for connection to the pipe lines, and regulation hose for connection to the machines. Racks for drill steel may also be included or may be carried on a separate truck or tender. Figure 22, *A*, shows a drill carriage and figure 22, *B*, a tender employed in driving an adit at Ojuela, Durango, Mexico.[77]

It will be noted that in this outfit only the drill bars are mounted on the carriage and the compressed-air connections are on the tender, which also carries a Coppus blower for ventilation.

Blasting may be done either with fuse and caps or electric detonators. Electrical firing has the advantages of safety and reduction in amount of smoke produced, and with delay detonators the proper sequence of firing and hence a better break is assured. Electric firing may be done from a power circuit or by use of blasting machines. When a power circuit is employed, the necessary switches must, of course, be provided. If the power circuit is a high-voltage one, step-down transformers also are necessary.

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[77] Savage, John P., Tunnel-Driving Methods Used at the Ojuela Unit of the Compañia Minera de Peñoles, S. A., Ojuela, Durango, Mexico: Bureau of Mines Inf. Circ. 6480, 1931, 8 pp.

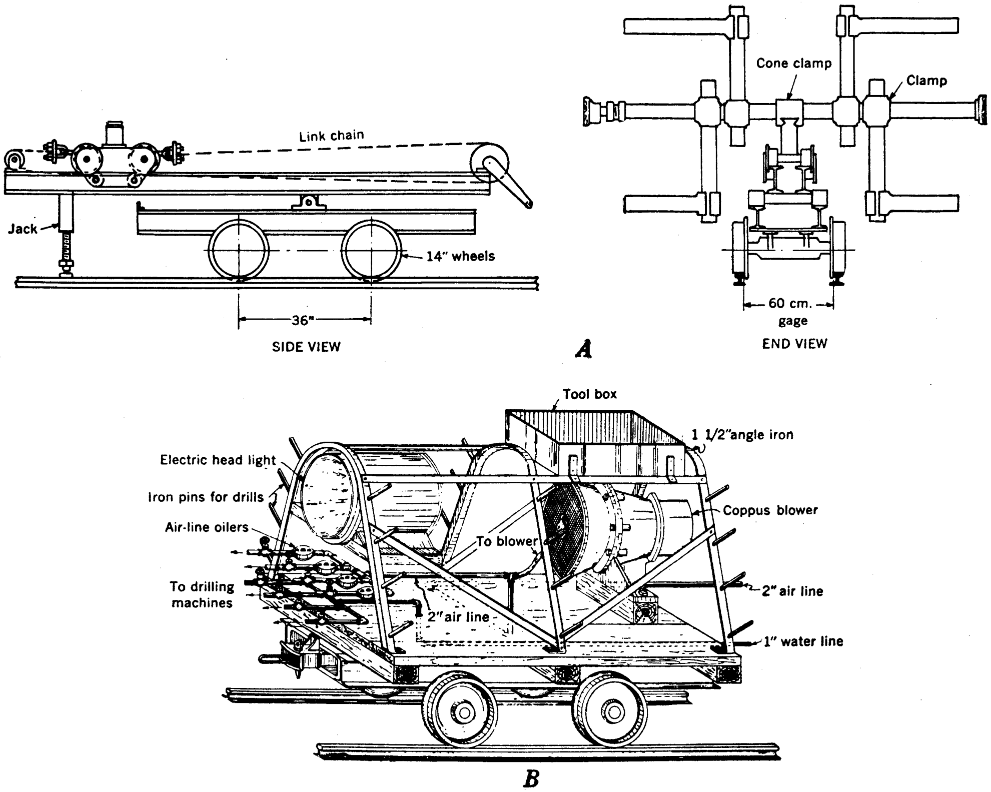
Link chain
Jack
14" wheels
36"
SIDE VIEW
A
Cone clamp
Clamp
60 cm. gage
END VIEW
Tool box
1 1/2" angle iron
Electric head light
Iron pins for drills
Air-line oilers
To blower
Coppus blower
To drilling machines
2" air line
2" air line
1" water line
B

FIGURE 22.—*A*, Drill carriage. *B*, Hezzelwood drill tender used at Ojuela, Durango, Mex.

When loading is by hand, picks, shovels, and sledges for breaking boulders are about all that is required. Mechanical loaders are of two general types—scrapers and shovel or dipper-type machines.

MECHANICAL LOADERS

Scrapers.—Scraper-loading has been discussed in an earlier Bureau of Mines publication.[78]

For use in driving headings, a scraper-loader consists of the following essential elements: A hoist, preferably double-drum, driven by electric motor or compressed air and mounted on a scraper slide or ramp; a scraper of the hoe, semihoe, or box type; pull and tail ropes; tail-rope sheave and means of anchoring it at the face; and a scraper slide mounted on wheels and terminating at the rear end in a boom, under which the rock cars are spotted for loading. The slide may, if desired, be mounted on caterpillars. Cars may be spotted and changed by hand, locomotive, or a car-puller.

Figure 23, *A*, shows a common type of scraper slide with the hoist mounted above it and a car spotted in loading position under the boom, and figure 23, *B*, is one with the hoist mounted below the slide. Figure 24 shows an outfit in which the slide and boom are mounted on a large car and the hoist on a tender at the rear. This equipment is adapted to straight headings having a cross section of 8 by 10 feet or larger; delays in loading occasioned by frequently changing cars, which are unavoidable with small cars, may be reduced to a minimum by the use of large cars. Figure 25 shows several types of scrapers used with loading slides.

Among the advantages of scraper loaders is their comparatively small first cost and low maintenance charges. Power costs are low, especially if electric drive is employed; primary power consumption with electric drive is only about one-eighth that with compressed-air drive. The largest single item of operating expense next to labor is renewal of wire ropes. Another advantage lies in the fact that only the scraper, a relatively inexpensive part of the equipment, need go near the face, which is an advantage in some ground where the back may fall. The hoist, scraper, and ropes can often be used interchangeably between development headings and stopes, thus reducing the total investment in loading equipment.

Although scrapers sometimes are used in small headings, ordinarily they will not save much as compared to hand loading in headings less than about 7 by 7 or 8 by 8 feet in cross section. For headings smaller than this, a small shovel-type loader is often preferable. For headings of this size and larger, scraper loading usually can show a saving in cost per foot and greater driving speed as compared to hand loading. By properly proportioning the size of the scraper and the capacity of the hoist to the size of the heading, scraper loading can be employed successfully in any size of heading above the minimum. If the proportion is right and proper facilities are provided for handling cars and eliminating delays, about 2 hours will be required to muck out a heading, regardless of its size.

With any type of mechanical loading, efficient operation depends on elimination of delays, which add to the time the machine must remain at the face and reduce the ratio of actual loading time to total time. This is particularly important in speed work, where each

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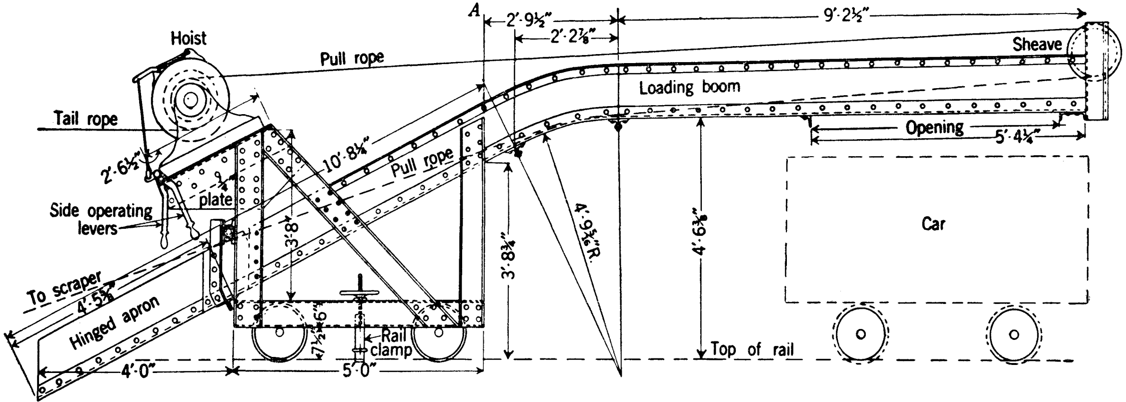
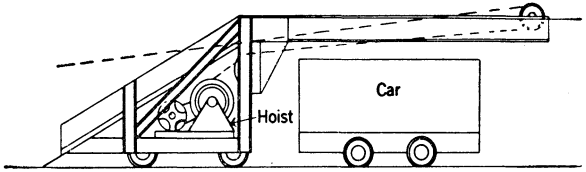
[78] Jackson, Chas. F., Underground Scraping Practice in Metal Mines: Bureau of Mines Manuscript Rept. 1, 1933, reprinted by Sullivan Machinery Co., 1933, 88 pp.


A
2'-9½"
2'-2⅞"
9'-2½"
Hoist
Pull rope
Sheave
Loading boom
Tail rope
2'-6½"
10'-8¾"
Pull rope
plate
Side operating levers
Opening
5'-4¼"
To scraper
4'-5⅝"
Hinged apron
3'-8"
3'-8¾"
4'-9⁵⁄₁₆"R.
4'-6⅜"
Car
7½"
6"
Rail clamp
Top of rail
4'-0"
5'-0"

PORTABLE SLIDE, HOIST MOUNTED ABOVE SLIDE

A



PORTABLE SLIDE, HOIST MOUNTED BELOW SLIDE

B

FIGURE 23.—Types of scraper slides.

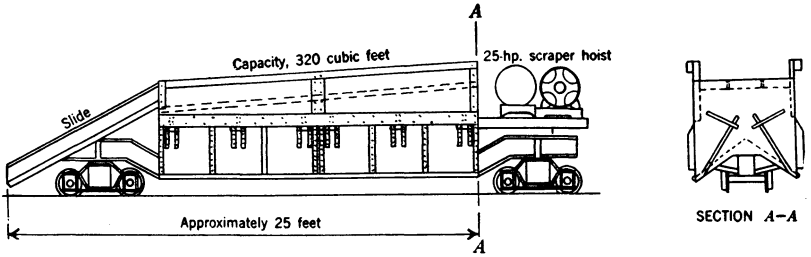

A
Capacity, 320 cubic feet
25-hp. scraper hoist
Slide
Approximately 25 feet
A
SECTION A–A

FIGURE 24.—One-car scraper-slide unit, Montreal mine, Wisconsin. (From Handbook of Scraper Mucking.)

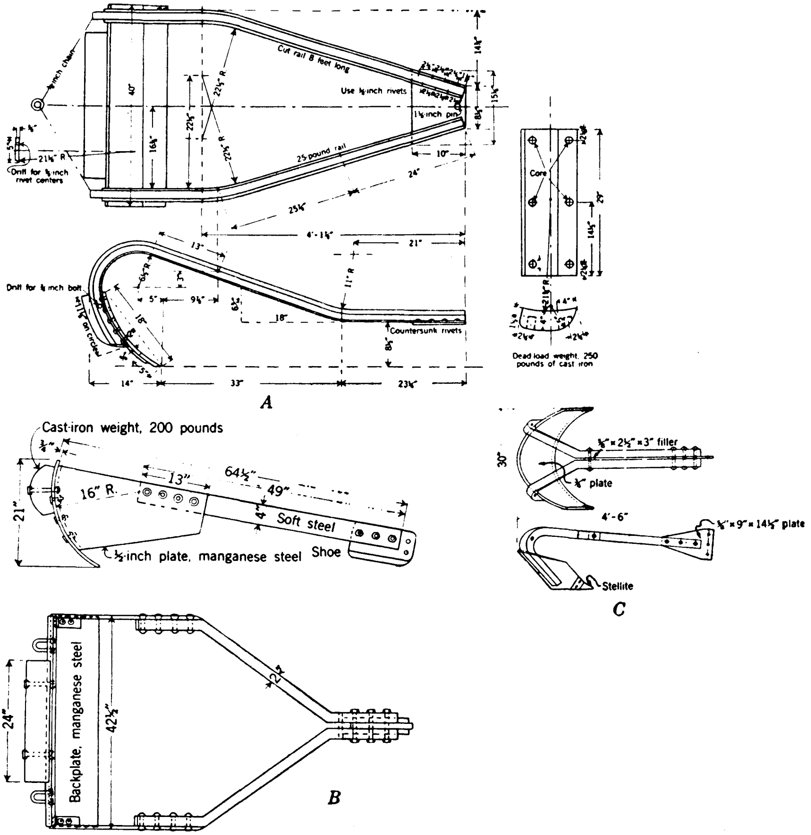

Cut rail 8 feet long
Use ⅝-inch rivets
1⅛-inch pin
25-pound rail
Drill for ⅝-inch rivet centers
Drill for ⅝-inch bolt
on circle
Countersunk rivets
Core
Dead load weight, 250 pounds of cast iron
40"
22½" R
10"
24"
25¼"
4'-1½"
21"
13"
18"
11" R
14"
33"
23½"
A
Cast-iron weight, 200 pounds
16" R
13"
64½"
49"
21"
Soft steel
½-inch plate, manganese steel
Shoe
Backplate, manganese steel
24"
42½"
B
30"
⅝"×2½"×3" filler
¾" plate
4'-6"
⅝"×9"×14½" plate
Stellite
C

FIGURE 25.—Three types of scrapers used with slides. *A*, Hoe type. *B*, Semihoe type. *C*, Crescent scraper.

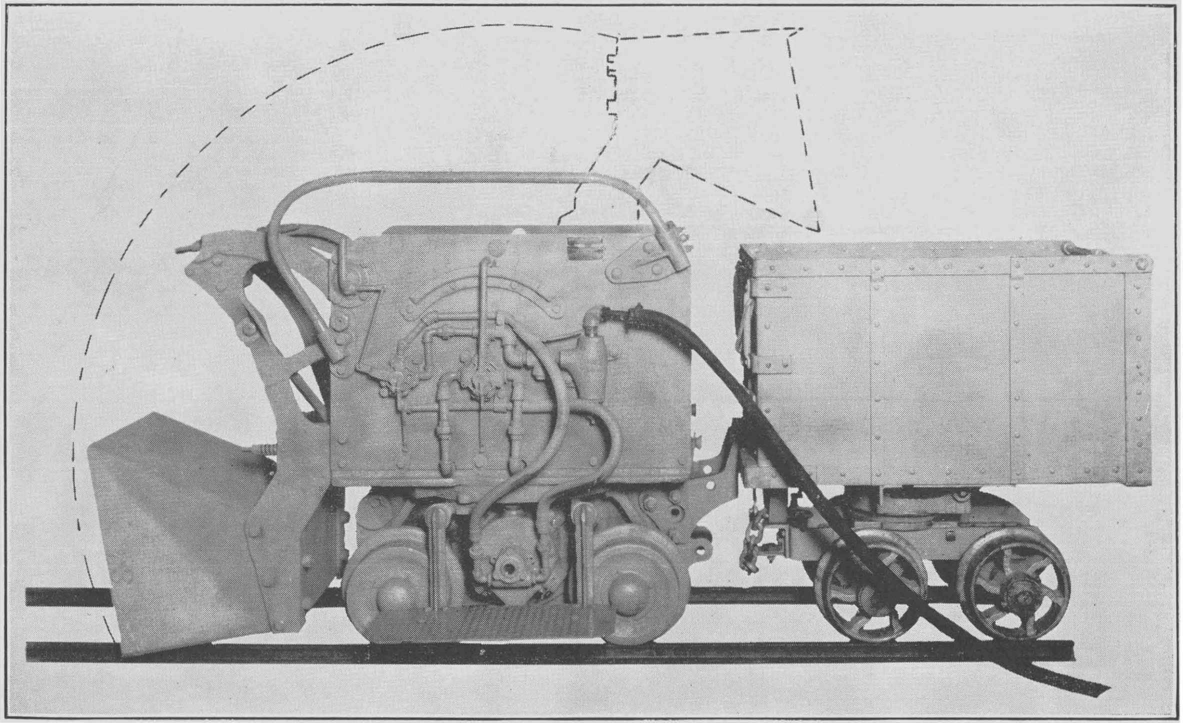


FIGURE 26.—Underground loader; dipper shovels directly into close-coupled car.

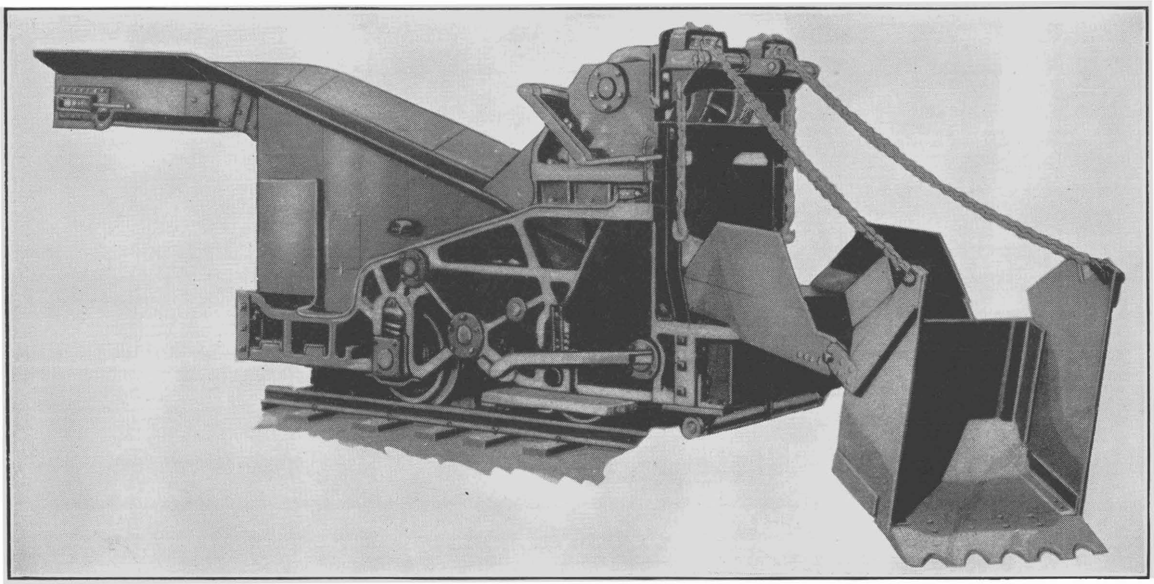


FIGURE 27.—Underground loader; dipper shovels onto conveyor.

cycle (drilling, blasting, mucking) follows the preceding one in quick succession.

Shovel loaders.—Shovel loaders employed in metal-mine work are of two general types—those that load directly into a car close-coupled behind the loader (fig. 26), and those that load onto a conveyor belt that delivers the rock to the car (fig. 27). They are generally driven by compressed air. The first type is most common in mine work and is particularly well adapted to headings smaller than 7 by 7 or 8 by 8 feet in cross section. Scraper loaders are competitors in larger headings, and the choice depends on local conditions or personal preference. The second type of shovel loader also competes with scrapers in larger headings but is generally not applicable to small headings. This and previous generalizations are based on personal observations of current practice and equipment, and it is realized that as they are written or before they are published new machines may be on the market that might necessitate modification of the statements made.

Shovel loaders commonly are mounted on wheels (fig. 27) and run on the mine track, and in types now most commonly employed the dipper is thrust into the muck pile by the tractive power of the machine. These loaders may be moved in and out of the heading under their own power or may be hauled by a mine locomotive. Scraper loaders usually must be hauled. Shovel loaders are also sometimes mounted on caterpillar crawlers (fig. 28). Another type of shovel loader is similar to shovels employed on surface, the thrust being transmitted to the dipper through a dipper stick or other thrust mechanism, which, when raised, may be swung to deliver the rock to one side of or behind the machine.

As this is written, a paper is in course of preparation on Shoveling Machines in Underground Metal Mines [79], to which the reader is referred for greater detail. This paper is based upon a study, extending over a year, of the use of shovel-type loaders in metal mines in a large number of districts in the Eastern, Central, and Western States.

Transportation equipment.—After the rock is loaded into cars by hand or mechanically, it usually is trammed to the dump in cars by hand or by a locomotive. Hand tramming is not, as a rule, economical for long distances. When the cars are loaded mechanically they are sometimes trammed a short distance by hand to a siding and from there hauled by locomotive. Small cars of 1 or 2 tons capacity are employed for hand tramming and cars of 1 to 6 tons or greater capacity when motor haulage is employed.

Large cars are preferred for mechanical loading, as they enable the loader to work more continuously. The one-car scraper-slide unit shown in figure 24 has already been mentioned. With this equipment, the entire unit is hauled to the dump when the car has been filled and loading is interrupted until it has been returned to the face. When ordinary cars are employed, the time occupied in switching is not necessarily all lost, as frequently the shovel (or scraper) can be employed in cleaning up, preparing the muck pile for loading, or in moving ahead.

Cars ordinarily are steel and of the types commonly employed in metal mines. They usually have roller bearings and may dump

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[79] Mosier, McHenry, and Steinmesch, J. H., Mechanical Shoveling in Underground Metal Mines: In press.

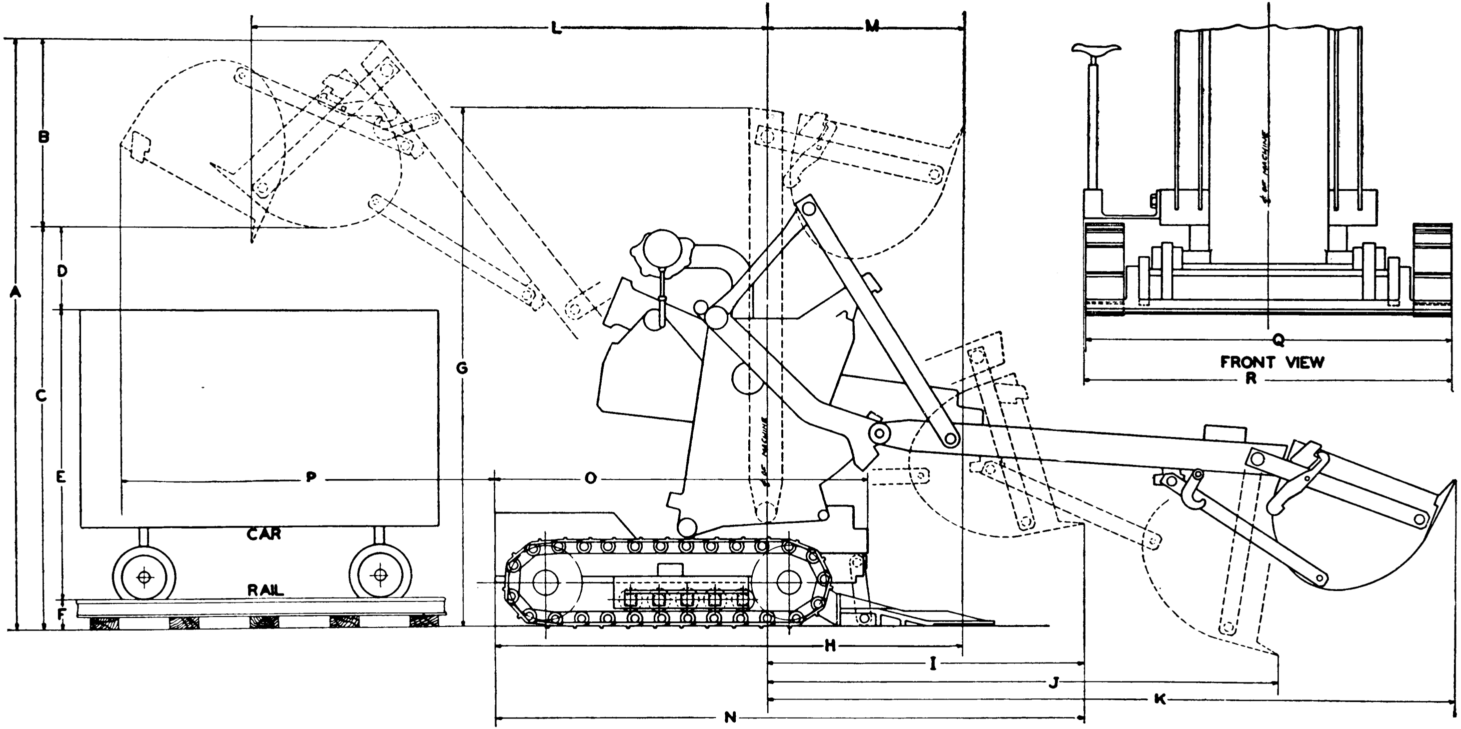
L
M
A
B
C
D
E
F
G
CAR
RAIL
P
O
H
I
J
K
N
Q
FRONT VIEW
R
℄ of machine
℄ of machine

FIGURE 28.—Underground loader on caterpillar crawl.

endwise or sidewise by tipping the body, or they may be of the bottom-discharge or gable-bottom type. The manner of dumping depends on facilities and conditions on the dump or may be determined by the type of car most readily available. Either trolley or storage-battery locomotives are suitable for haulage in adits and tunnels. Gasoline-driven locomotives should not be employed because of the danger from exhaust fumes.

Track is ordinary steel rail of proper weight for the service to which it is put or is to be put later and is laid on the adit grade. For locomotive haulage one-fourth to one-half percent upgrade to the face has been found most satisfactory. If the principal function of the adit is for drainage rather than haulage, steeper grades may be preferable.

To permit running cars and shovel loaders close to the face, short sections of track can be made up with fish plates for connection to the end of the permanent track. For quick change of cars behind a mechanical loader, movable switches and turnouts are convenient. Figure 29 shows the arrangement used in driving at Ojuela. An arrangement for lifting the car off the track and to one side has been employed satisfactorily to pass cars on a single track. In headings wide enough for double track, passing arrangements readily can be provided.

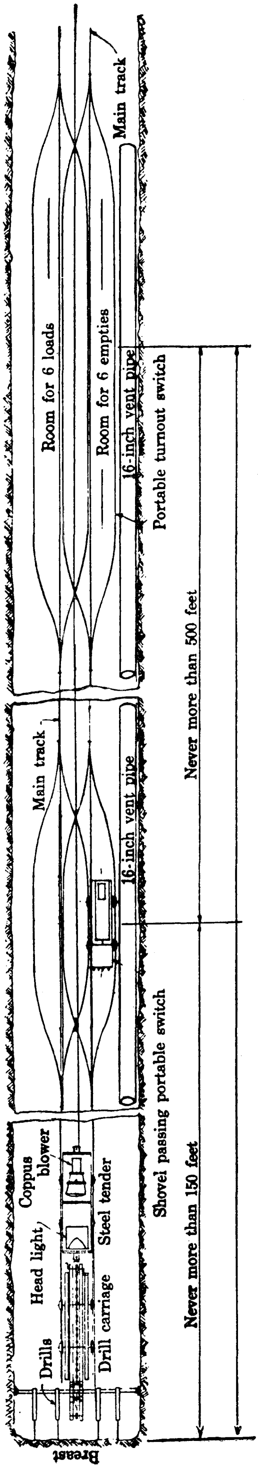
Main track
Room for 6 loads
Room for 6 empties
16-inch vent pipe
Portable turnout switch
Never more than 500 feet
Main track
16-inch vent pipe
Shovel passing portable switch
Never more than 150 feet
Coppus blower
Head light
Steel tender
Drills
Drill carriage
Breast

FIGURE 29.—Arrangement of switches for changing cars behind loader.

Rubber-tired trucks without track have been employed for transporting broken rock, and it is likely that in the future loading onto belt or pan conveyors extending from the face to the dump may come into quite common use in tunnel driving.

Ventilating equipment.—Ventilation of the face is important, particularly in long adits and high-speed driving. If natural ventilation only is employed, the time interval that must be allowed between blasting and return to the face for mucking becomes increasingly longer as the length of the adit increases, which obviously reduces the speed that otherwise might be attained. It has long been customary to turn on the compressed-air line and blow out the face after blasting and to depend to some extent, at least, on the exhaust from rock drills or loading machines to provide ventilation while men are working at the face.

Although this may be fairly satisfactory in short headings, it is an expensive method from the standpoint of power consumption in any case, and in long headings is not as satisfactory as an independent low-pressure ventilation system.

The most positive and efficient system is an independent low-pressure one with a reversible fan at the entrance delivering air to the face through a steel pipe. Collapsible tubing in place of the steel pipe is widely used when it is desired only to blow air to the face and is very handy for this purpose as an extension from the end of the pipe that is usually kept some distance back from the face to prevent damage from blasting. Thus, while drilling or mucking, the tubing can be attached to carry air right to the face, and just before blasting it can be rolled back out of the way.

In some instances return to the face after blasting can be expedited by exhausting through the pipe for a period and then blowing by reversing the fan. There is some difference of opinion about this, but if the pipe can be kept near enough to the face the fumes may be exhausted quickly while the workmen are entering, through fresh air drawn in to displace that exhausted through the pipe; whereas in blowing, some of the fumes simply will be churned about at the face for some time while the rest are being blown out the adit, and the men entering must pass through them at some place along the heading. Obviously, collapsible tubing cannot be used for exhausting.

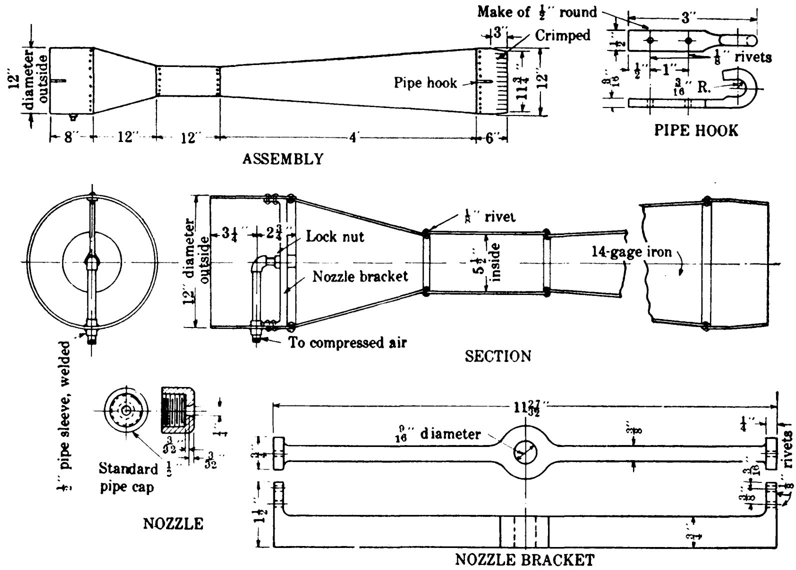
Make of ½'' round
3''
Crimped
3''
⅛'' rivets
½''
1''
3/16'' R.
PIPE HOOK
12'' diameter outside
Pipe hook
11½''
12''
8''
12''
12''
4'
6''
ASSEMBLY
⅛'' rivet
3¼''
2¾''
Lock nut
12'' diameter outside
Nozzle bracket
5½'' inside
14-gage iron
To compressed air
SECTION
¼'' pipe sleeve, welded
Standard ⅛'' pipe cap
NOZZLE
11 27/32''
9/16'' diameter
¼''
⅛'' rivets
NOZZLE BRACKET

FIGURE 30.—Venturi type of compressed-air injector used for auxiliary ventilation in the United Verde mine. Arizona.

Ventilators of the injector type have been employed to a considerable extent. They are cheaply constructed and easily installed but waste so much power that they are used only in temporary or emergency auxiliary service [80] (such as in driving adits or tunnels). Air jets and injectors take many forms, from nozzles designed to screw onto a compressed-air hose to well-designed venturi types (fig. 30). The nozzle is connected to the compressed-air line, as shown in the illustration. The efficiency of even the best venturi types is usually less than 7 percent for the jet and 45 percent for production of compressed air, or 3 percent over-all.

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[80] McElroy, G. E., Engineering Factors in the Ventilation of Metal Mines: Bureau of Mines Bull. 385, 1935, pp. 133–134.

METHODS OF DRIVING

It is not proposed to begin a long discussion of practices employed in driving mine headings that have been described in detail in an earlier Bureau of Mines publication.[81] Methods employed in driving adits are virtually the same as those used in drifts and crosscuts, numerous descriptions of which are given in the series of information circulars on mining methods and costs published by the Bureau beginning in 1929 and continuing to the present.

In driving large aqueducts, railroad and vehicular tunnels, and the like, special technique often is employed, such as carrying a low heading at the top ahead of the bottom bench, which is taken up later, or driving a small pilot tunnel that is later enlarged to finished size. In mine work, the face usually is advanced as a single full-size heading, though modifications of this may be employed where unusual conditions, such as running, swelling, or sloughing ground, occur. Thus, it may be necessary to keep timber close to the face and spile ahead, or to drive a narrow vertical slot on both sides and over the top of the face, put in posts, cap, and lagging, and then remove the core.

Generally, adits may be advanced by drilling and blasting regular rounds, which often can be standardized for the particular conditions of each job. The principle upon which standard types of rounds are based is to drill cut-holes, which when blasted pull out a small plug or wedge of rock, providing free faces to which surrounding holes, timed to go slightly later, will break. In ground at all tight it is axiomatic that if the cut-holes fail to break out the cut the other holes will fail to break well unless unusual slips or seams happen to cause a freak break.

The principal types of rounds are those employing a V or wedge-shaped cut; a "pyramid" cut that pulls a pyramid or cone-shaped block near the center of the face; a "Michigan" or "burnt" cut that pulls a cylindrical core near the center of the face; and the "swing" or "slabbing" cut. By whatever names these cuts may be called locally, they are of these types or merely variations thereof. The V cut may be drilled to pull a horizontal wedge extending across the face from side to side or a vertical one from top to bottom. Such a wedge is usually across or up and down near the center line of the face but may be above, below, or to one side. Sometimes the wedge is along one rib or along the bottom, when it may be termed a "draw" cut or, in the latter case, a "toe" cut.

In the V cut two rows of holes are drilled at angles to the face such that opposite holes meet at the bottom or nearly so (fig. 31, *A*). In the pyramid cut the holes are drilled at angles to meet, or nearly so, at a common point to permit concentration of the explosive charge as much as possible at the bottom of the cut (fig. 31, *B*). In this cut there may be 3 to 8 (usually 4) holes all drilled toward a common point. In the burnt cut a number of holes are drilled straight in (normal to the face), as in figure 31, *C*. In the round depicted, the two center holes (in black) receive no explosive charge, but the cut holes *1* and easers *2* are loaded and shot. The center holes provide space and lines of weakness for the rock to break to. This round is particularly suited to small headings where the rock is tough and the ground tight. Figure 31, *D*, shows a slabbing round sometimes employed in wide headings. In this round the order of firing is such that the short holes go first, breaking out a triangular-shaped (in plan) block toward the free face, thus providing a new free face for the next holes to break to, and so on successively to the last and longest holes.

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[81] Gardner, E. D., Drilling and Blasting in Metal-Mine Drifts and Crosscuts: Bureau of Mines Bull. 311, 1929, 170 pp.

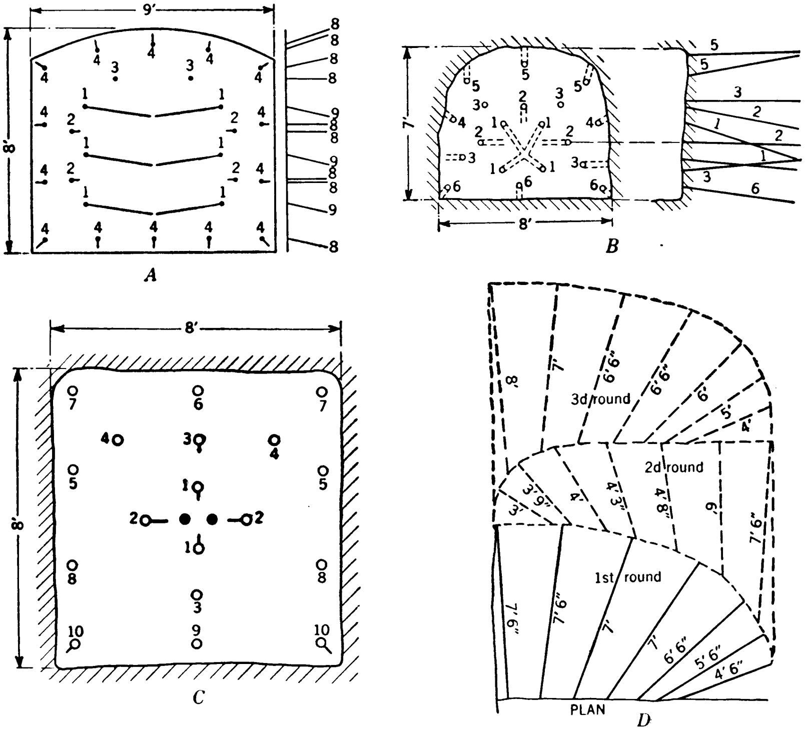


FIGURE 31.—*A*, V cut. *B*, Pyramid cut. *C*, Burnt cut. *D*, Slabbing or swing-round.

In cut-hole rounds it is sometimes the practice, where it is difficult to break the round, first to load and blast only the cut-holes, return to the face, and, if the cuts do not break, reload them and then load the rest of the holes and blast the entire round. When this is done, it is obvious that good ventilation is required to permit prompt return to the face after the cuts are first blasted.

In any type of round it is essential that the order of firing be such that holes progressively farther from the cut are blasted in succession.

Many adits, particularly if driven in country rock, require no artificial support, or if so, only in certain sections. The back often will stand better without artificial support if it is arched in the center. Some ground will stand unsupported for a time and will then begin to slough. In this instance, timbering can follow some distance behind the face without interruption of the drilling-blasting-mucking cycle.

In worse ground it may be necessary to carry the timbering close to the face and put in a set after each round. Figure 32, *A*, shows a three-piece set of timber and figure 32, *B*, an arch set.

Sometimes adits that are in bad ground and are to be in use for a long time are concreted, the concrete being put in after the adit has been completed or in sections, one at a time, as it advances. Figure 32, *C*, shows a typical concrete lining.

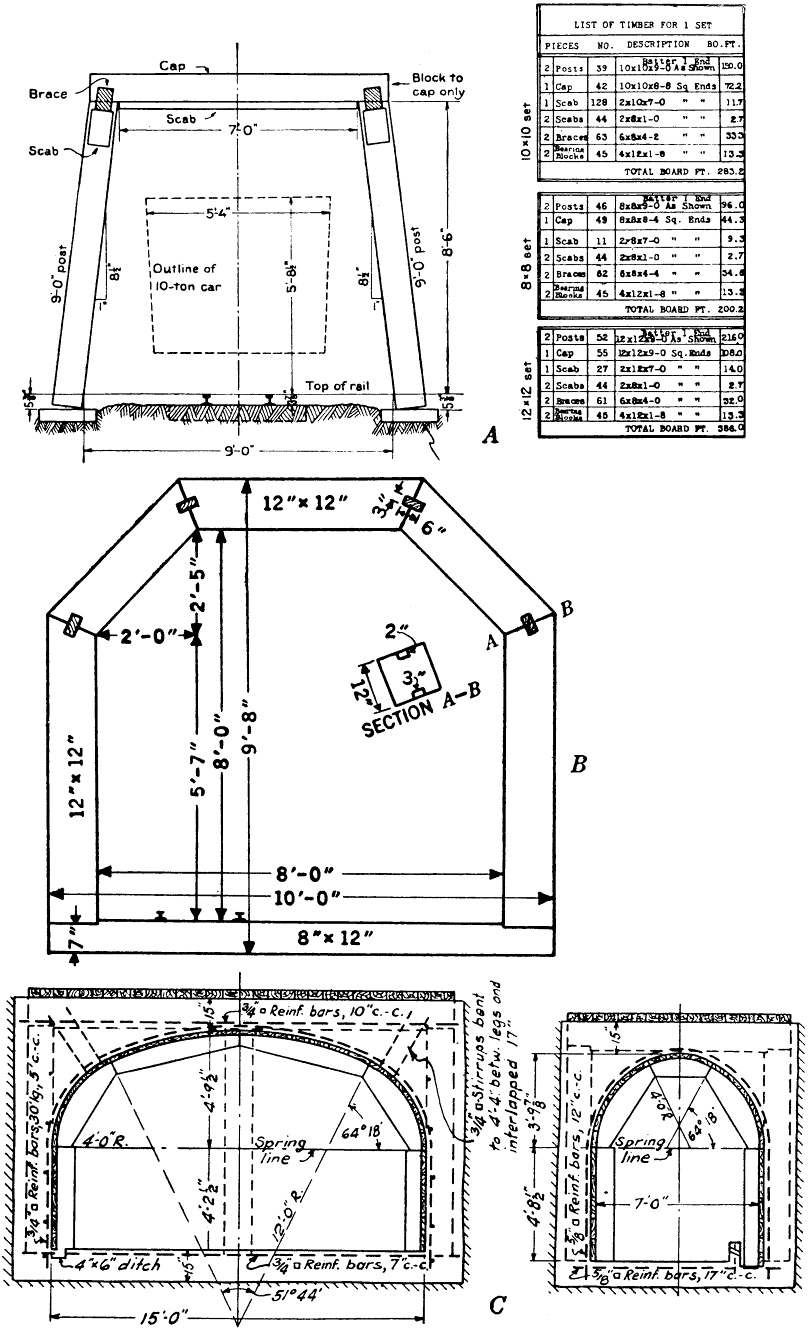
Organization of driving operations is important, especially in three-shift speed work. Where speed is not particularly desired and hand loading is employed on regular shifts, work is generally scheduled so that drilling and blasting are done on one shift and mucking on the next, with a time interval between shifts adequate to allow the smoke to clear. The crew in this case is usually divided into two classifications, miners or machinemen and muckers. In the United States it has been customary for many years to do this type of work on contract; that is, at a straight price per foot of advance. This price usually covers labor only or labor and explosives, but the company guarantees at least minimum prevailing wages. The price may be in the form of a schedule that provides for variations in the rate, depending on the character of the rock, and extra payment for timbering, track laying, and other extras. An alternative is payment according to a bonus system, which provides standard wages for a certain minimum advance per month or per pay period and a graduated increase for footage advanced in excess of the minimum.

Where speed is required, three shifts usually are employed, which may be consecutive or overlapping. In continuous work it is obvious that positive ventilation must be provided to reduce delays after blasting to a minimum. Each operation—drilling, blasting, or shoveling—is completed in the shortest time possible, and each shift usually is composed of a crew capable of performing the work of any part of the cycle under a boss or shift leader. Usually, a bonus system of payment is employed, whereby additional wages are paid for footage that exceeds a fixed minimum per pay period or per month. There are variations in organization of speed work too numerous to be discussed in detail here. When mechanical mucking is in use, there may be a separate crew that comes on after each blast and works split shifts, or the loading machine may be operated by the regular shift crews. Track laying usually is done by the regular crew as part of the contract price, and timbering generally is done by the regular crew when timber must be set after each round is mucked out, or by a special timber crew where timbering follows some distance behind the face.

COST OF DRIVING

Costs vary considerably with the size of the heading, drillability of the rock and relative ease with which it breaks, amount and nature of ground support required, speed of driving, equipment employed, labor and supply cost, and other local conditions.

These considerations are obvious for the most part and require no particular discussion except with regard to speed of driving. Up to a certain rate, which varies with conditions, greater speed usually results in savings in cost per foot by reason of reduction of unit overhead or general charges. Beyond this critical rate, the cost per foot will tend to increase. High-speed work is necessarily nicely organized,


Cap
Brace
Block to cap only
Scab
Scab
7'-0"
5'-4"
Outline of 10-ton car
9'-0" post
9'-0" post
8½"
8½"
1"
1"
5'-8½"
8'-6"
Top of rail
5⅜"
5⅜"
9'-0"
A
12"× 12"
3"
6"
2'-5"
2'-0"
2"
3"
12"
SECTION A-B
A
B
12"× 12"
5'-7"
8'-0"
9'-8"
8'-0"
10'-0"
7"
8"× 12"
B
15"
¾"⌀ Reinf. bars, 10" c.-c.
¾"⌀ Stirrups bent to 4'-4" betw. legs and interlapped 17"
¾"⌀ Reinf. bars 30" lg., 5" c.-c.
4'-0"R.
4'-9½"
Spring line
64°18'
4'-2½"
12'-0"R.
4"×6" ditch
15"
¾"⌀ Reinf. bars, 7" c.-c.
51°44'
15'-0"
15"
3'-9⅜"
4'-8½"
⅝"⌀ Reinf. bars, 12" c.-c.
4'-0"
64°18'
Spring line
7'-0"
⅝"⌀ Reinf. bars, 17" c.-c.
C

	LIST OF TIMBER FOR 1 SET				
	PIECES		NO.	DESCRIPTION	BO. FT.
10×10 set	2	Posts	39	10x10x9-0 As Shown Batter 1 End	150.0
	1	Cap	42	10x10x8-8 Sq Ends	72.2
	1	Scab	128	2x10x7-0 " "	11.7
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	63	6x8x4-2 " "	33.3
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	283.2
8×8 set	2	Posts	46	8x8x9-0 As Shown Batter 1 End	96.0
	1	Cap	49	8x8x8-4 Sq. Ends	44.3
	1	Scab	11	2x8x7-0 " "	9.3
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	62	6x8x4-4 " "	34.6
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	200.2
12×12 set	2	Posts	52	12x12x9-0 As Shown Batter 1 End	216.0
	1	Cap	55	12x12x9-0 Sq. Ends	108.0
	1	Scab	27	2x12x7-0 " "	14.0
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	61	6x8x4-0 " "	32.0
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	386.0

FIGURE 32.—*A*, Three-piece timber set. *B*, Arch timber set. *C*, Conventional concrete arch designs for double- and single-track haulageways. (After Robert W. Thomas, Am. Inst. Min. and Met. Eng. Tech Pub. 730.)

and any disruption in the regular cycle of operations is reflected in increased costs. Thus, for example, where speed is the prime objective, consumption of explosives is likely to be high, due to pulling rounds of more than the most economical length and liberality in amount used per round to insure that the round will break. However, even though costs per foot may be higher, increased speed may actually effect an over-all money saving in the mine operation by advancing the date when ore development and extraction can be begun.

Table 13 gives a few typical examples of costs of driving adits that for the most part are untimbered. Timbering or concreting obviously will add considerably to the cost. At Ray, Ariz. (fig. 32, *C*), cost of concreting was $22 to $25.24 per cubic yard of concrete placed.[82]

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[82] Thomas, Robert W., Concreting Drifts at Ray Mines Division of Kennecott Copper Corporation: Am. Inst. Min. and Met. Eng. Tech. Pub. 730, 1936, 14 pp.

TABLE 13.—*Example of adit-driving costs*

Name and year	Size of rock section, feet	Kind of rock	Method of drilling	Method of mucking	Timbering	Speed of driving	Direct costs per foot					
							Labor and supervision	Compressed-air drills and steel	Power	Explosives	Other materials and general expense	Total
Ojuela, Mexico,[1] 1927–1928.	8½ by 9	300 ft. caliche, 3,257 ft. calcareous shales, 1,850 ft. limestone.	28-hole pyramid cut. (Fig. 33.) 4 machines and drill carriage.	Shovel-type mechanical loader.	None except in caliche.	714 ft. per month. Average cycle 5 hrs. 37 min.	$10.33	$3.48	$0.84	$3.77	$2.945	$21.365
Halkyn, Wales,[2] July 1–15, 1933.	10 by 8	Hard, crystalline limestone.	36 holes including 8 cuts; 2 machines on crossbars.	Scraper slide	Practically none.	495 feet in 168 hours.	[3] 13.375	[3] 1.984	[3] 0.066	6.095	[3] 2.537	[3] 24.057
Park-Utah, Utah,[4] about 1929.	8 by 8	Medium hard	2 machines on crossbar.	Hand	About ⅓ timbered.	Muck and drill simultaneously.	16.56	2.01	----------	2.44	2.83	23.84
Britannia, B. C.[5], prior to 1935.	10 by 12	Very hard and blocky to very soft and schistose.	4-drill carriage and steel car, 29-hole pyramid-cut round.	Scraper slide	None	6 feet per shift complete cycle.	----------	----------	----------	----------	----------	[6] 25.83
Colony Coal Co., Colo.,[7] 1930.	11 by 8	Shales and sandstone.	Undercut with coal cutter. 6-hole round drilled with coal drill.	Duck-bill loader attached to shaking conveyor.	Timbered with 3-piece sets and lagged.	7.31 feet per day, 2 shifts.	5.55	1.13	Not given.	.48	1.54 (timber)	8.70
Sheep Creek, Alaska,[8] prior to 1914.	10 by 8	474 ft. slide rock; 4,009 ft. greenstone; 4,224 ft. slate; 1,085 ft. metagabbro.	Short (4-ft.) rounds. 4 machines on crossbars. 21 to 23 holes (29 exceptionally) center V-cut.	Hand	Only in loose ground near portal.	Av. 16 months, 544.2 ft. per month. Max. 661 feet.	18.86	2.64	2.53 (power and compressed air.)	4.47	2.58	[9] 31.08
B. C. Nickel,[10] Sept. 1934.	8½ by 10	Granodiorite and hornblendite.	3 machines, pyramid cut, 31 holes.	Shovel-type loader.	Little timbering required.	505 feet during month.	13.51	4.38	----------	5.94	2.79	[11] 26.62
Suyoc, P. I.,[12] 1937.	11 by 11, inside timbers.	Gabbro	4 machines on jumbo, 30 holes, 8 ft. round.	Conway loader.	None in portion covered by cost figures.	1,088 June	----------	----------	----------	----------	----------	[13] 12.78
						1,327 July	----------	----------	----------	----------	----------	[14] 16.45

[1] Savage, John P., Tunnel-Driving Methods Used at the Ojuela Unit of the Compañia Minera de Peñoles, S. A., Ojuela, Durango, Mexico: Bureau of Mines Inf. Circ. 6480, 1931, 8 pp.

[2] Allan, J. C., Twelve-Mile Bore to Drain Halkyn Lead District [North Wales]: Eng. and Min. Jour., vol. 135, No. 6, June 1934, pp. 253–255.

[3] To make comparable with costs at other mines listed, cost of hoisting, $1.033, not included.

[4] Hewitt, E. W., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Bureau of Mines Inf. Circ. 6290, 1930, 18 pp.

[5] Brennan, C. V., The Mining Operation at the Property of the Britannia Mining and Smelting Co., Ltd., Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6815, 1934, 36 pp.

[6] Includes proportion of all construction and equipment expense and capital charges for compressor, steel shop, 100-pound track, and trolley wire.

[7] Bryson, W. D., A Mechanically Driven Level Rock Tunnel: Bureau of Mines, Inf. Circ. 6598, 1932, 4 pp.

[8] Engineering and Mining Journal, Driving the Sheep Creek Tunnel: Oct. 17, 1914, pp. 693–698.

[9] Includes depreciation on mining tools, $1.29, and loss on boarding house, $1.17.

[10] Merrett, E. J., Tunneling at B. C. Nickel Mines: Canada Min. and Met. Bull., 297, January 1937, pp. 11–16.

[11] Cost of 4,546 feet of tunnel, $23.76.

[12] Harman, John W., Fast Tunneling in the Philippines: Eng. and Min. Jour., March 1938, pp. 53–56.

[13] Breaking, $7.92; mucking, $0.84; tramming, $1.03; ventilation, $0.01; engineering, $0.10; general overhead, $2.88.

[14] Breaking, $9.32; mucking, $0.87; tramming, $0.28; ventilation, $0.05; engineering, $0.08; general overhead, $5.85.

EQUIPMENT, METHODS, AND COSTS OF SHAFT SINKING

The technical literature is replete with descriptions of and data on shaft-sinking methods and costs. The subject of shaft sinking has been discussed at some length in an earlier Bureau of Mines bulletin.[83] A number of shaft-sinking operations are described in detail in the series of information circulars on mining methods and costs, to some of which reference will be made later. Hence, the present discussion will be confined to the broader aspects of the subject.

Small shallow shafts in isolated districts often are sunk by hand, hand windlasses or whims being used for hoisting the muck in small buckets, and such methods frequently are employed to depths of 10 to 25 feet or more in starting larger shafts.

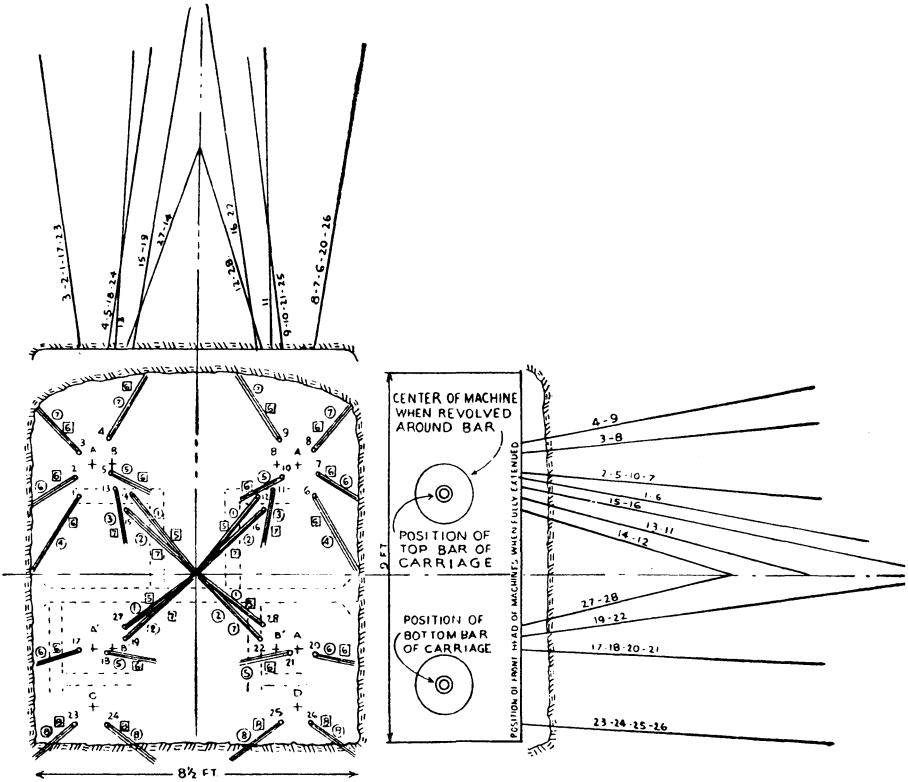
3-2-1-17-23
4-5-18-24
13
15-19
27-14
12-28
16-22
11
9-10-21-25
8-7-6-20-26
CENTER OF MACHINE WHEN REVOLVED AROUND BAR
POSITION OF TOP BAR OF CARRIAGE
POSITION OF BOTTOM BAR OF CARRIAGE
POSITION OF FRONT HEAD OF MACHINES WHEN FULLY EXTENDED
2 FT
4-9
3-8
2-5-10-7
1-6
15-16
13-11
14-12
27-28
19-22
17-18-20-21
23-24-25-26
A
B
A'
B'
C
D
8½ FT

FIGURE 33.—Standard 28-hole round.

During the past 20 or 30 years there have been decided improvements in the technique, equipment, and practices employed in shaft sinking, both as applied under normal hard-rock conditions and for sinking through wet, caving, or running ground.

SHAFT-SINKING EQUIPMENT

The usual equipment for sinking at metal mines may be grouped roughly under eight classifications—drilling, blasting, mucking, hoisting, pumping, ventilation, power plant, and miscellaneous equipment, not including general shops and necessary building structures. A great deal of this equipment, such as air compressors and power-plant and shop equipment, will be already available if the shaft is being sunk at an operating mine, and general equipment of this nature is not discussed here.

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[83] Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bureau of Mines Bull. 357, 1932, 104 pp.

Drilling equipment comprises air compressors, rock drills and accessories such as hose and fittings, drill steel and steel-sharpening or grinding equipment, air and water pipes, and manifolds or headers. Rock drills for shaft work usually are of the hand-held or "plugger" type weighing 35 to 80 pounds, although heavier machines mounted on cross bars sometimes are employed for drilling very hard ground in long narrow shafts. Modern practice calls for hollow drill steel and wet drilling, using conventional types of forged bits or detachable bits.

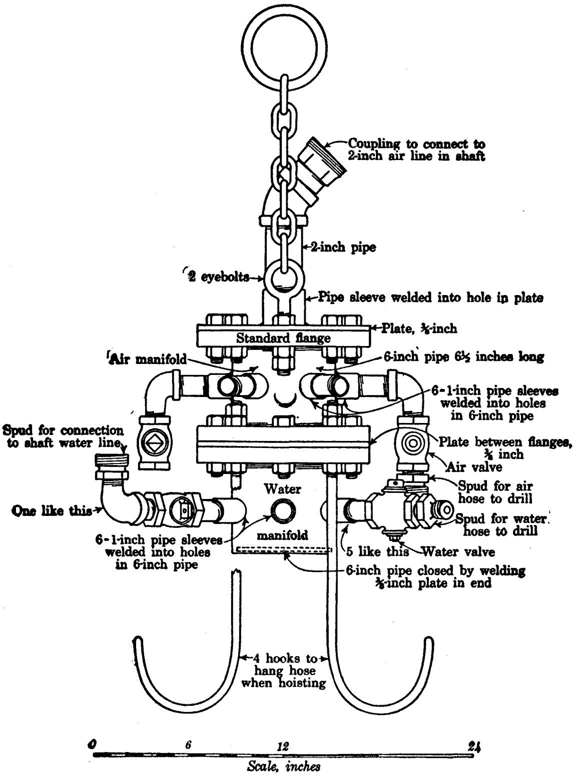
Coupling to connect to 2-inch air line in shaft
2-inch pipe
2 eyebolts
Pipe sleeve welded into hole in plate
Plate, ⅜-inch
Standard flange
Air manifold
6-inch pipe 6½ inches long
6-1-inch pipe sleeves welded into holes in 6-inch pipe
Spud for connection to shaft water line
Plate between flanges, ⅜ inch
Air valve
Spud for air hose to drill
Water
One like this
Spud for water hose to drill
manifold
6-1-inch pipe sleeves welded into holes in 6-inch pipe
5 like this
Water valve
6-inch pipe closed by welding ⅜-inch plate in end
4 hooks to hang hose when hoisting
0
6
12
24
Scale, inches

FIGURE 34.—Typical air and water header or manifold.

In the larger shafts, where several drills are operated at a time, air hose, line-oilers, and water-hose connections to the drills usually are attached to a manifold or header. In the largest shafts there may be separate air and water headers. The header is connected to the air and drilling-water pipes in the shaft by heavy hose. It is usually raised and lowered quickly after and before drilling operations by a tugger hoist mounted on the shaft timbers 30 to 50 feet or more above the bottom of the shaft, or by chain blocks, or it may be hoisted to the surface when not in use. These headers may be made up of pipes and fittings, or of pipes, fittings, and special castings assembled in a variety of ways, one of which is shown in figure 34 and which is a combination air and water header. Some headers are also provided with hooks on which to hang the rock drills.

Although fuse and caps are still employed for blasting in small shafts, modern practice requires electric blasting for greater safety, for reducing the number of misfires, and for more precise timing of the order in which the holes will go. Electric blasting necessitates the use of either blasting machines or switches and wiring for connection to a power circuit. With blasting machines the leg wires of the detonators are connected in series, and with power-circuit firing they are connected in parallel.

In either instance leading or main wires usually are wound on a reel for convenience in lowering to the bottom of the shaft from the surface or from the end of the fixed wiring in the shaft or in an upper shaft station. The lead and fixed wiring should be of approved type, insulated to withstand water, and of ample capacity for the length of transmission and the current to be carried. Permanent wires should be attached to insulating spools or other standard suspension and should not be wound around nails or secured by other makeshifts.

Blasting shields for protection of the shaft timber against flying rock may be classed arbitrarily as part of the blasting equipment, although they serve other purposes of equal usefulness, such as protection for the men at the shaft bottom against falling rock or material and as staging on which they may work when installing timber. Figure 35 shows a common form of blasting shield. They may be constructed of standard structural-steel shapes and perforated plates, of timber, or of a combination of steel and timber, with openings for passage of the bucket, skip or cage, for ventilating and other pipes, and for men. For convenience in lowering, the set may be suspended by cables or from chain blocks at each corner, or both. The shield should be as light as possible, consistent with strength and rigidity. In small shafts it is common practice to use a false set bolted to the under side of the last permanent shaft set for protection of the latter instead of a more elaborate blasting set of the type described above.

Mucking usually is done by hand with ordinary pointed shovels, picks, and sledges for breaking large boulders, although at least one successful attempt to employ scraper loading has been reported.[84] The conventional method of mucking is to shovel by hand into a sinking bucket or small skip. In large, deep shafts it is necessary to hoist in buckets or skips of relatively large capacity to reduce the hoisting time. To avoid the use of high buckets, which are difficult to shovel into, and to reduce mucking delays, a low loading "pan" or "scoop" has been used successfully in a number of shafts. The pan is filled by hand shoveling and is then lifted a few feet by means of a small hoist set on the permanent shaft timbers above and dumped into a skip (fig. 36) or into a car on a special sinking cage. This scheme has been credited with reducing mucking time by as much as 20 percent at some mines, although operators disagree as to its efficacy, some preferring direct mucking into skips or buckets.

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[84] Mendelsohn, Albert, Mining Methods and Costs at the Champion Mine, Painesdale, Mich.: Bureau of Mines Inf. Circ. 6515, 1931, 16 pp.

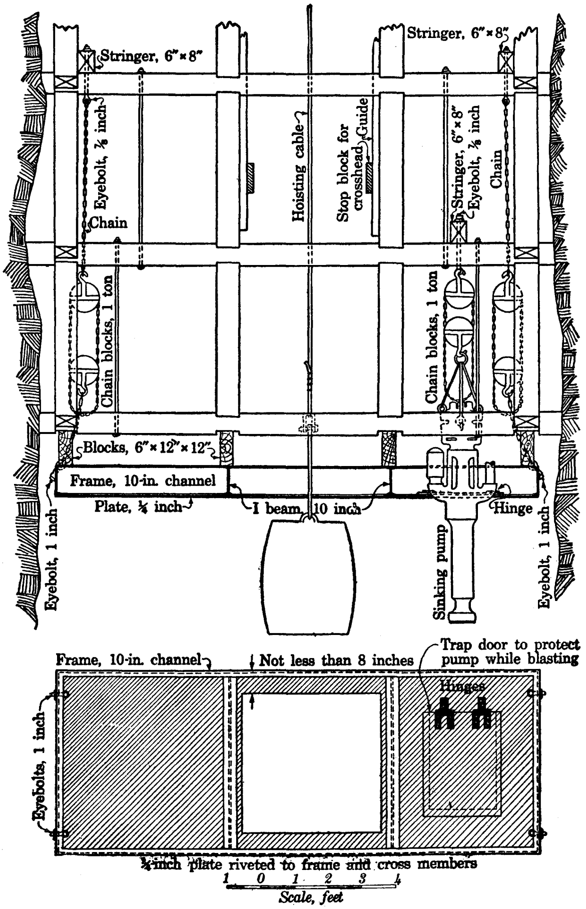
Stringer, 6"×8"
Stringer, 6"×8"
Eyebolt, ⅞ inch
Chain
Hoisting cable
Stop block for crosshead
Guide
Stringer, 6"×8"
Eyebolt, ⅞ inch
Chain
Chain blocks, 1 ton
Chain blocks, 1 ton
Blocks, 6"×12"×12"
Frame, 10-in. channel
Plate, ¼ inch
I beam, 10 inch
Hinge
Eyebolt, 1 inch
Sinking pump
Eyebolt, 1 inch
Frame, 10-in. channel
Not less than 8 inches
Trap door to protect pump while blasting
Hinges
Eyebolts, 1 inch
¼-inch plate riveted to frame and cross members
1 0 1 2 3 4
Scale, feet

FIGURE 35.—Common form of blasting set.

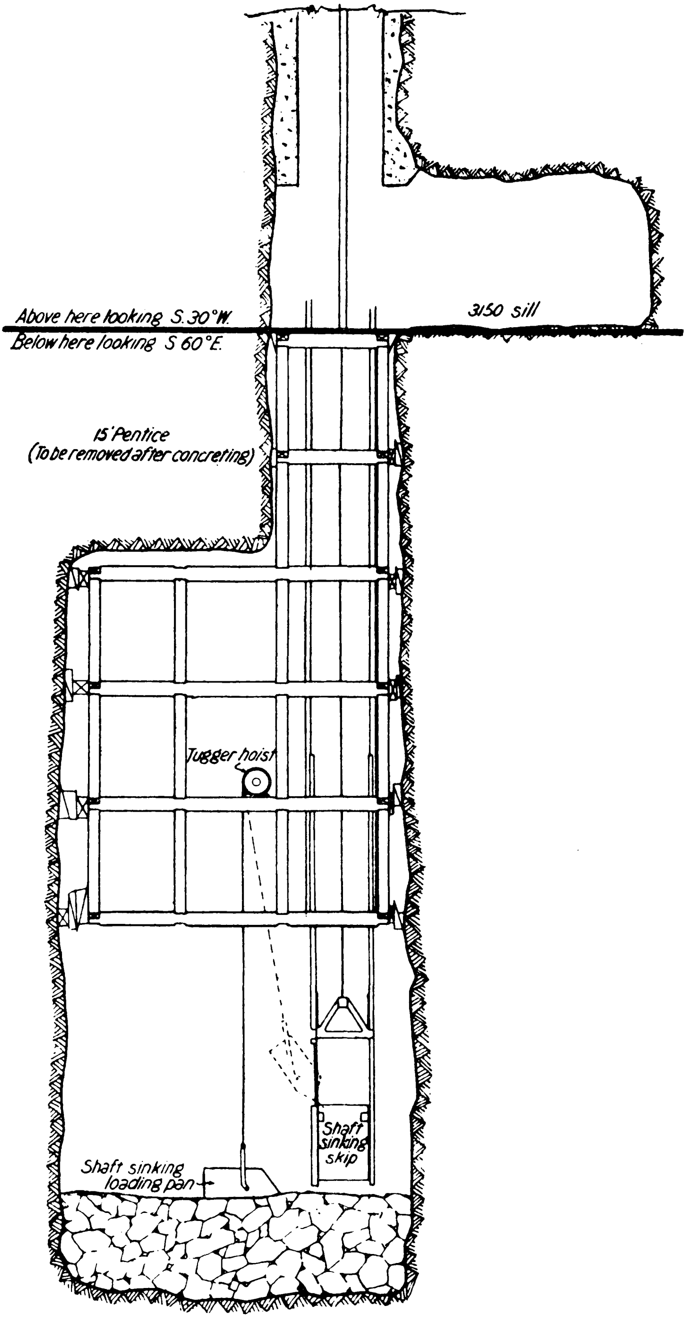
Above here looking S. 30°W.
Below here looking S 60°E.
3150 sill
15' Pentice
(To be removed after concreting)
Tugger hoist
Shaft sinking skip
Shaft sinking loading pan

FIGURE 36.—Loading pan and shaft-sinking skip.

Hoisting equipment includes, in addition to the auxiliary hoists already referred to, buckets, crossheads, skips and cages, hoisting rope, signal system between the hoistman and the shaft bottom and shaft landing, the hoist, and the head-frame structure, dumping arrangements, and head sheaves.

Sinking buckets usually range in capacity from 10 or 12 to 30 cubic feet, or from ½ to 1½ tons, though sometimes their capacity is as small as 750 pounds and rarely are they as large as 2 or 3 tons. They may be constructed as shown in figure 37, with the bail attached at the top and a ring or a chain and ball attached to the bottom for overturning at the dump, or with the bail attached on the sides just below the center of gravity. The latter type is kept from overturning in the shaft by a ring on the bail, which slips over a lug extending above the rim of the bucket.

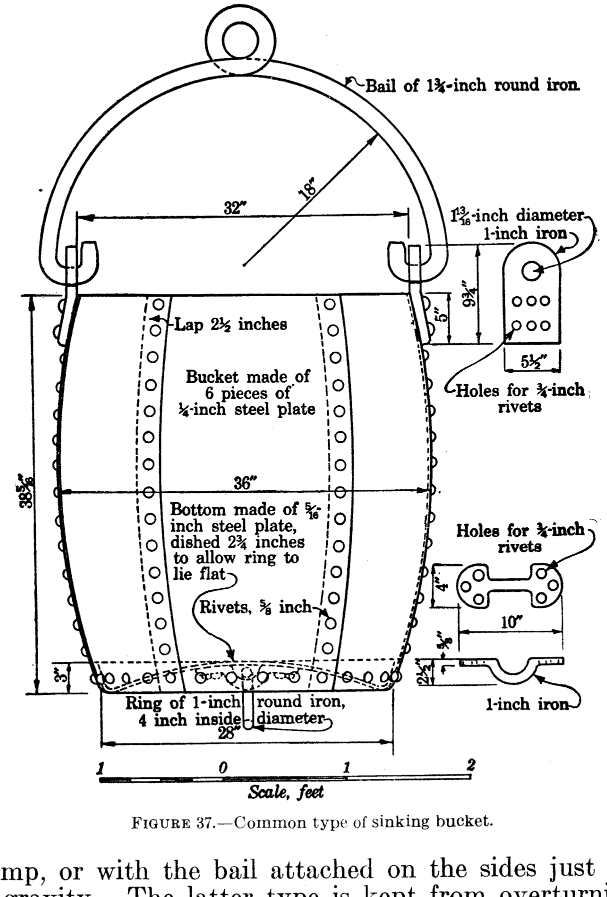
Bail of 1¾-inch round iron
18"
32"
1$\frac{13}{16}$-inch diameter
1-inch iron
9¾"
5"
5½"
Holes for ¾-inch rivets
Lap 2½ inches
Bucket made of 6 pieces of ¼-inch steel plate
38$\frac{5}{16}$"
36"
Bottom made of $\frac{5}{16}$-inch steel plate, dished 2¾ inches to allow ring to lie flat
Rivets, ⅝ inch
Holes for ¾-inch rivets
4"
10"
⅝"
2½"
3"
1-inch iron
Ring of 1-inch round iron, 4 inch inside diameter
28"
1
0
1
2
Scale, feet

FIGURE 37.—Common type of sinking bucket.

In sinking to depths of 200 or 300 feet, buckets without crossheads are sometimes used. Thus, in the Tri-State zinc and lead district it has been the practice for years to sink single-compartment shafts about 5 by 7 feet in cross section, using 1,200-pound buckets or "cans" attached to "nonspinning" ropes, and without a crosshead. However, except for very shallow depths, crossheads should be used to prevent spinning and swinging of the bucket against the sides of the

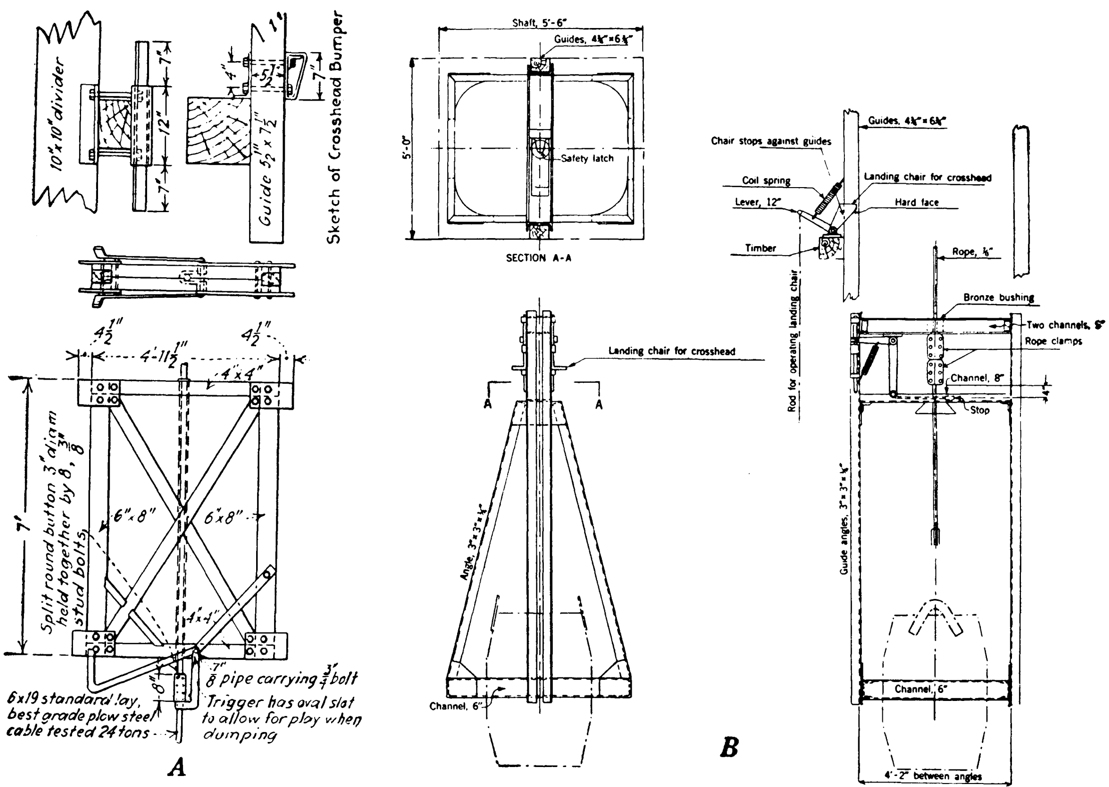
10"x10" divider
7"
12"
7"
4"
$5\frac{1}{2}$"
7"
1"
Guide $5\frac{1}{2}$" x $7\frac{1}{2}$"
Sketch of Crosshead Bumper
$4\frac{1}{2}$"
$4'\text{-}11\frac{1}{2}$"
$4\frac{1}{2}$"
4"x4"
7'
Split round button 3" diam held together by 8, $\frac{3}{8}$" stud bolts
6"x8"
6"x8"
4"x4"
7"
8"
$\frac{7}{8}$ pipe carrying $\frac{3}{4}$ bolt
6x19 standard lay, best grade plow steel cable tested 24 tons
Trigger has oval slot to allow for play when dumping
A
Shaft, 5'-6"
Guides, $4\frac{1}{4}$"x$6\frac{1}{4}$"
5'-0"
Safety latch
SECTION A-A
Landing chair for crosshead
A
A
Angle, 3"x3"x$\frac{1}{4}$"
Channel, 6"
B
Guides, $4\frac{1}{4}$"x$6\frac{1}{4}$"
Chair stops against guides
Landing chair for crosshead
Coil spring
Hard face
Lever, 12"
Timber
Rod for operating landing chair
Rope, $\frac{7}{8}$"
Bronze bushing
Two channels, 5"
Rope clamps
Channel, 8"
4"
Stop
Guide angles, 3"x3"x$\frac{1}{4}$"
Channel, 6"
4'-2" between angles

FIGURE 38.—Types of crossheads. *A*, Simple truss crosshead. *B*, Special crosshead.

shaft. A single deep beam with a shoe on each end to fit over the guides sometimes is used but may cause accidents by jamming between the guides while the bucket is in motion. A simple truss structure, the depth of which is at least two-thirds the distance between guides, is often employed (fig. 38, *A*)[85]; or, for greater safety, a construction may be used that embodies a bonnet or hood and safety latches and safety dogs designed so that the bucket and crosshead will stay together and the dogs will grip the shaft guides and hold the crosshead if the rope breaks. Figure 38, *B*, shows a special type of crosshead used at some mines in Ontario, which prevents the bucket from swinging and is equipped with a safety latch that prevents the bucket from dropping below the crosshead if the latter should catch and hang up in the shaft. Figure 39 shows a safety crosshead equipped with safety dogs to grip the guides if the rope should break.[86]

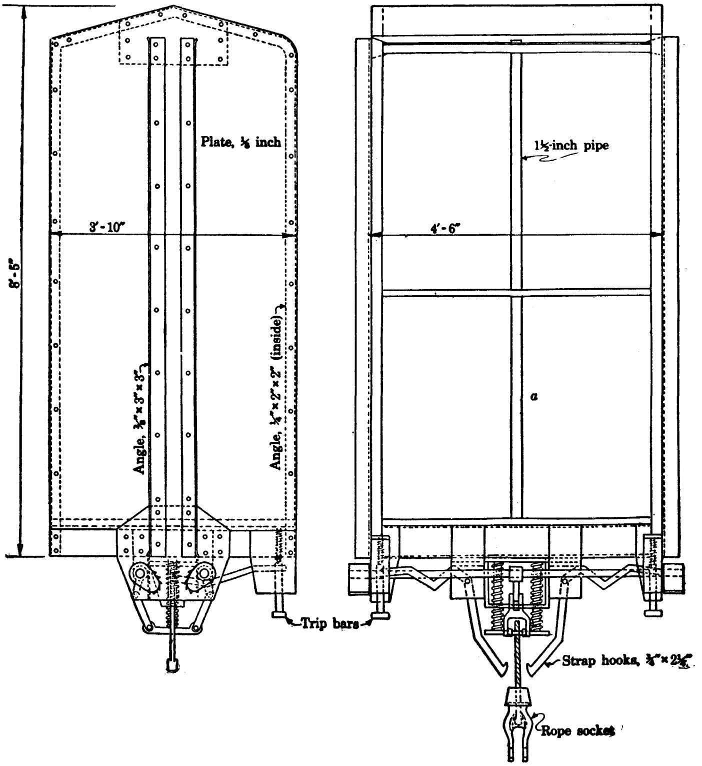

FIGURE 39.—Safety crosshead used in sinking Eureka (Utah) shaft.

When skips or mine cages are employed for shaft sinking, they are equipped with long extension runners or shoes that engage the wooden shaft guides or their extensions below the timbers when the car is in loading position at the shaft bottom (fig. 40).

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[85] Sawyer, D. L., Shaft-Sinking Practice at Copper Mountain: Eng. and Min. Jour., vol. 130, No. 4, Aug. 23, 1930, p. 166.

[86] Engineering and Mining Journal, Safety Crosshead Reduces Accidents and Attendant Expenses: Vol. 128, No. 20, Nov. 16, 1929, p. 780.

In sinking inclined shafts, buckets may be employed if the inclination is steep, when they may either run on skids or on a trolley (fig. 41); for flat inclinations, skips or cars running on rails are used.

For signaling the hoistman, the usual practice is to employ a wire pull-cord, which operates a gong or knocker, although electric systems that signal direct from the shaft bottom are sometimes used. A combination of the two methods is not uncommon in deep shafts; the shaft men signal by pull-cord to a landing or station higher up, whence the signal is relayed to the surface by an attendant by means of an electrical system.

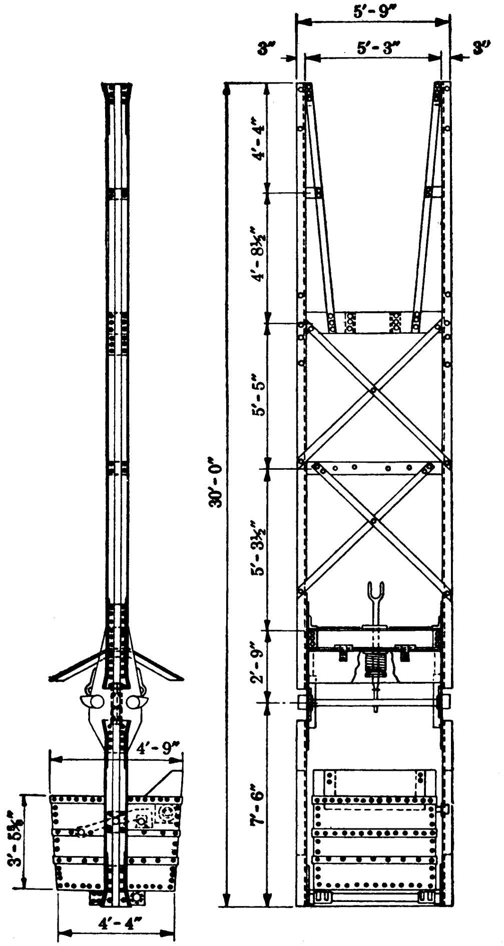

FIGURE 40.—Shaft-sinking skip used at the No. 3 Frood shaft, Ontario.

Geared hoists usually are employed that will handle the desired load at a speed ranging from 300 feet per minute in small shafts and for shallow depths to 1,500 feet per minute for large deep shafts. Steam, compressed-air, gasoline-engine, or electric hoists may be employed for shaft sinking. Electric drive is usually preferable if

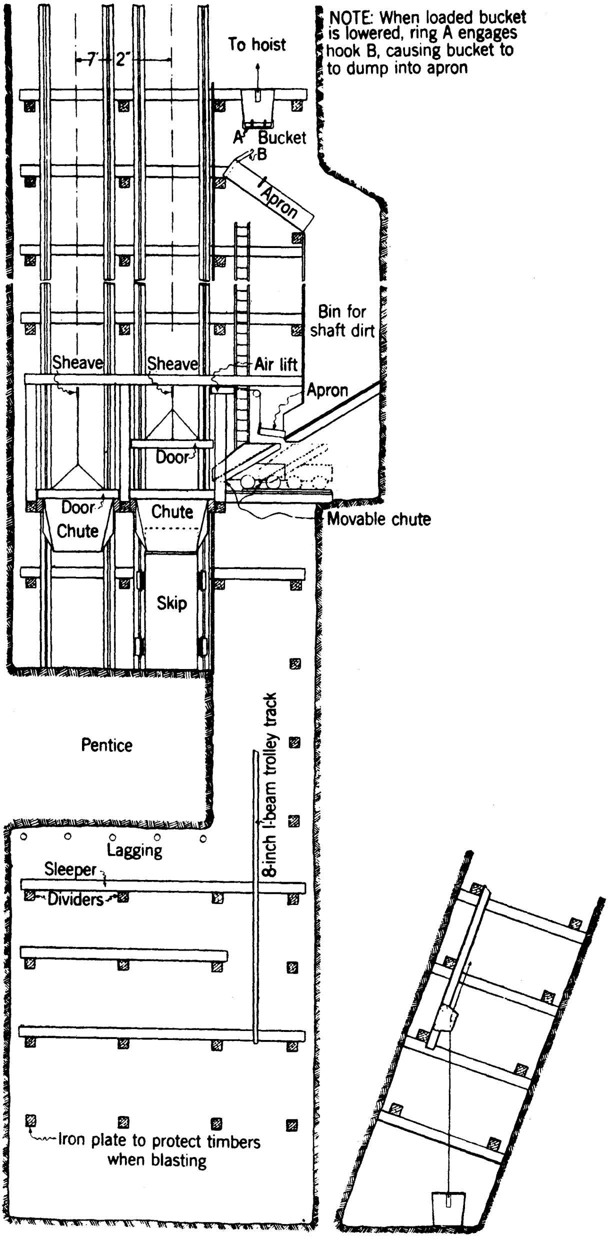
NOTE: When loaded bucket is lowered, ring A engages hook B, causing bucket to to dump into apron
7'
2"
To hoist
A
Bucket
B
Apron
Bin for shaft dirt
Sheave
Sheave
Air lift
Apron
Door
Door
Chute
Chute
Movable chute
Skip
Pentice
8-inch I-beam trolley track
Lagging
Sleeper
Dividers
Iron plate to protect timbers when blasting

FIGURE 41.—Shaft-sinking method, Champion mine, Michigan, showing trolley for hoisting bucket.

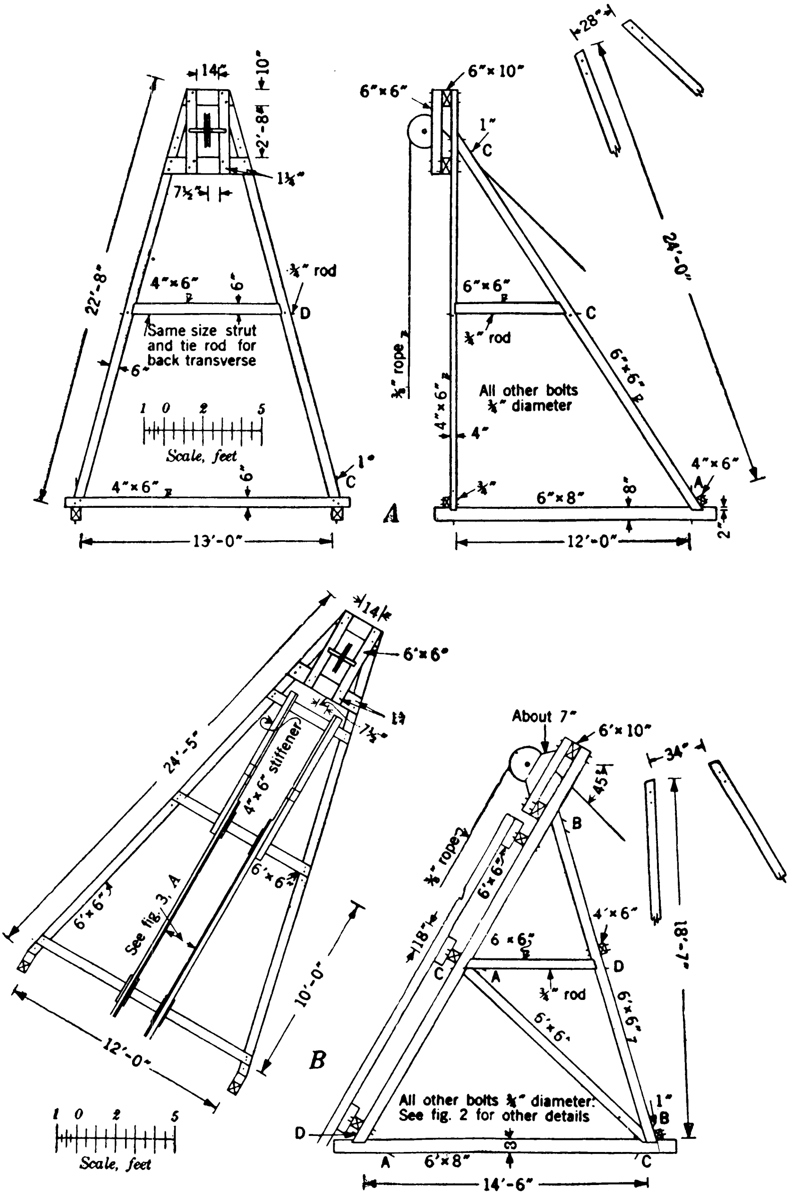
14"
10"
2'-8"
1¼"
7½"
22'-8"
4"×6"
6"
¾" rod
D
Same size strut and tie rod for back transverse
6"
1 0 2 5
Scale, feet
4"×6"
6"
1"
C
13'-0"
A
6"×6"
6"×10"
1"
C
¾" rope
6"×6"
C
¾" rod
4"×6"
4"
All other bolts ¾" diameter
6"×6"
¾"
6"×8"
8"
4"×6"
A
2"
12'-0"
28"
24'-0"
14"
6"×6"
1½"
7½"
24'-5"
4"×6" stiffener
6"×6"
See fig. 3, A
6"×6"
10'-0"
12'-0"
B
1 0 2 5
Scale, feet
About 7"
6"×10"
45°
B
34"
¾" rope
6"×6"
18"
4"×6"
6×6"
C
A
D
¾" rod
6"×6"
6"×6"
All other bolts ¾" diameter. See fig. 2 for other details
1"
B
D
8"
A
6"×8"
C
14'-6"
18'-7"

FIGURE 42.—Small sinking head frames (dumping arrangements omitted). *A*, For vertical shaft. *B*, For inclined shaft.

power is available at moderate cost. Slow-speed hoists sometimes are employed for starting the shaft and are replaced by higher-speed hoists after the shaft has attained considerable depth. For shallow depths and small shafts single-drum hoists usually serve the purpose, two buckets being used, one of which is loaded while the other is being hoisted, dumped, and lowered. As greater depths are attained, double-drum hoists sometimes are employed to speed mucking operations, three buckets being used, one of which is loaded while the other two are being hoisted and lowered in balance.

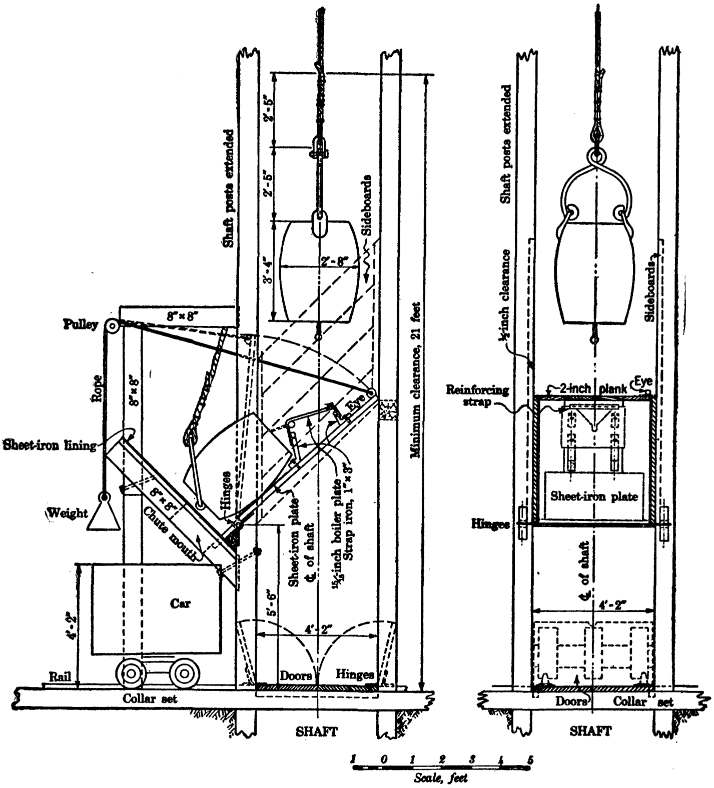
Shaft posts extended
2'-5"
2'-5"
3'-4"
2'-8"
Sideboards
Minimum clearance, 21 feet
Pulley
8"×8"
Rope
8"×8"
Sheet-iron lining
Weight
8"×8"
Chute mouth
Hinges
Eye
Sheet-iron plate
℄ of shaft
15/16-inch boiler plate
Strap iron, 1"×3"
Car
4'-2"
5'-6"
4'-2"
Rail
Doors
Hinges
Collar set
SHAFT
Shaft posts extended
½-inch clearance
Sideboards
Reinforcing strap
2-inch plank
Eye
Sheet-iron plate
Hinges
℄ of shaft
4'-2"
Doors
Collar set
SHAFT
1 0 1 2 3 4 5
Scale, feet

FIGURE 43.—Arrangement for dumping buckets at the surface used by the Walter Fitch, Jr., Co., Eureka, Utah.

Head frames may be made of either timber or steel. Sometimes the permanent shaft head frame is erected before sinking is begun or as soon as the shaft has been sunk a few feet and the permanent collar has been installed. More often, a temporary sinking head frame is used. This is usually of such dimensions that the permanent head frame may be erected around and over it later without interruption of operations. Sinking head frames may be of either the A-frame or the four-post type and are equipped with suitable arrangements for dumping buckets or skips or for caging and uncaging cars, as the case may be. Figure 42 illustrates small sinking head frames for hoisting in buckets, and figure 43 shows dumping arrangements in a vertical head frame. These may be varied in different ways to suit conditions, personal preference, or legal requirements. Note the hinged doors over the shaft collar, which should always be provided as a safety measure. Sometimes the head frame is made higher to allow for a rock bin below the dump. The design of small wooden head frames has been discussed in an earlier Bureau of Mines publication.[87]

Pumping equipment includes the pumps proper, suction and discharge hose, pipe, pipe fittings, and pipe hangers or supports. Formerly, steam-driven pumps were quite common in shaft-sinking work, but they are seldom employed now. Their use caused great inconvenience by reason of the heat from the steam and exhaust pipes; wet shafts were sometimes crowded with hot pipes and water columns, and the hazard of a breaking steam line was always present. The use of compressed-air-driven pumps is much to be preferred and is now common practice. Electrically driven pumps of both the multistage centrifugal and plunger types that are suitable for sinking have come into use and offer certain advantages due to greater flexibility and ease of handling.

In small shafts, sinker pumps that are suspended by a bridle in a vertical position may be required; but where there is enough room, ordinary horizontal pumps are preferable, as they are lighter, cheaper, and, in the smaller sizes, easier to handle. In shaft-sinking the water contains a large amount of coarse and fine grit, and for this reason plunger pumps are better suited to the work than are standard centrifugal pumps. In recent years new rubber compositions and rubber cements have been developed that have been utilized for centrifugal sand-pump impellers in ore-dressing plants, and they might, perhaps, be applied to centrifugal pumps in shaft sinking.

A suction hose wrapped with rope for protection and a section of hose connecting the pump discharge and the lower end of the water column in the shaft provide flexibility in moving the pumps out of the way when blasting and make it unnecessary to add short sections of pipe to the bottom of the column with every cycle of the sinking operations. The water column is usually standard pipe, which should have flanged couplings, Victaulic connections, or other couplings that do not have to be screwed on and off. When a large flow of water must be handled, heavy pumps may be mounted on a framework provided with shoes to run on guides extending to the shaft bottom, the pump and framework being lowered and raised by an auxiliary hoist.

It frequently happens in shaft sinking that the inflow of water is concentrated at one or more comparatively short sections. In this instance it is economical to excavate water rings around the shaft below the heavy flow to catch the water. A pump may then be installed permanently at this point to lift the water to the surface. Water rings may also be convenient to collect water pumped from the bottom of the shaft and thus reduce the discharge head on the sinking pump.

Ventilation during shaft sinking is furnished by rock-drill exhaust, compressed-air-driven blowers, air nozzles or injectors similar to those mentioned in the discussion of adits, or fans. In continuous three-shift work it is desirable to remove the smoke quickly from the shaft after blasting, and a centrifugal fan on the surface is the most efficient equipment for this purpose. The fan may deliver air to the bottom of the shaft through pipe or flexible tubing 8 to 12 inches in diameter or may exhaust smoke and gases from the bottom of the shaft through ventilating pipe extending close to the shaft bottom. Pipe is preferable for several reasons; flexible tubing can only be used for blowing, whereas an exhaust system is often more efficient; and when pipe is used, the fan, if reversible, may be operated either blowing or exhausting, as circumstances dictate. Moreover, tubing tends to kink or collapse; in long lines considerable power is expended in keeping the tube open, and the frictional loss is high. With pipe and a reversible fan, smoke may be removed quickly by exhausting followed by reversal of the fan for blowing fresh air to the bottom of the shaft while men are working. When blowing, a few lengths of flexible tubing may be attached to the lower end of the pipe to permit directing the air where it will be most effective. Ventilating pipe usually is galvanized iron, plain or spiral-riveted, of 16- to 20-gage iron, depending on the diameter of the pipe. Wood-stave pipe has been employed in tunnel work, where, in at least one instance, it was found to be more serviceable than iron pipe and easier to keep tight at the joints.

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[87] Staley, W. W., The Design of Small Wooden Head Frames: Bureau of Mines Inf. Circ. 6943, 1937, 37 pp.

Under miscellaneous equipment may be listed that required for framing shaft timber, for pouring concrete linings (concrete mixer, steel or wooden forms, and pipe, hose, or buckets for lowering concrete), and equipment for special shaft-sinking jobs. The latter category includes equipment for sinking through quicksand by the shield method or by the caisson method under air pressure, equipment for sealing off heavy flows of water in fractured rock by the cementation process, and that required for employing the freezing process. Contracting companies making a specialty of the caisson method or the freezing process are usually employed for such work.

The shield method has been used for sinking short distances through wet sand. The equipment is illustrated by figure 44, which is a cross section through one side or one end of the shaft. The shoe with trailing plates extending upward and overlapping the last regular shaft set is jacked ahead and the sand is excavated inside. This shoe extends around the shaft on all four sides, the members being rigidly bolted together.

In the cementation process, quick-setting cement or cement and sand are forced by means of a plunger pump through pipes inserted in holes drilled outward from the shaft into the fractured ground.

Occasionally a stiff-leg derrick with hoist and clamshell or orange-peel bucket has been used for sinking through overburden to bedrock.

Eaton [88] presented an excellent discussion of shaft-sinking equipment in 1932.

SHAFT-SINKING METHODS AND PRACTICES

Conventional shaft-sinking practice involves the performance of a cycle of different operations—drilling and blasting, removal of smoke and cleaning of fly-rock lodged on overhead timbers, mucking and hoisting of the broken rock, and timbering. The latter (*a*) may be the last operation of the cycle, (*b*) may be carried on during drilling and mucking, provided it is done from a substantial blasting shield or bulkhead so that the men below may be protected from injury by falling objects, or (*c*) may be done in sections while other work is stopped; that is, if the walls of the shaft stand well, several sets may be placed at one time following several drilling-blasting-mucking cycles during which no timbering is done.

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[88] Eaton, Lucien, Shaft-Sinking Equipment—I: Eng. and Min. Jour., vol. 133, No. 3, March 1932, pp. 165–170; also Shaft-Sinking Equipment—II: Vol. 133, No. 4, April 1932, pp. 219–224.

Good organization of the work is required, especially where speed is desired. The crew is usually made up of miners in charge of a working shift leader, who are capable of performing the work of any part of the cycle so that each shift as it comes on duty takes up the work where the preceding shift left off. This applies particularly to continuous three-shift sinking. There are variations of this procedure, and often, especially if timbering is concurrent with drilling and mucking, a special timbering crew may be employed. In other instances three separate crews are employed—one for drilling and blasting, one for mucking and hoisting, and one for timbering. This system may be used to advantage where speed is not of prime importance and short enough rounds are drilled so that the mucking crew can clean out the entire round in one shift.

In any event, the crews are made up of shaftmen and surface or topmen. Best results are obtained with shaftmen trained especially in shaft work; many good stope, drift, or raise miners do not make good shaftmen. The top crew comprises the hoist engineers, toplanders, the mechanics in charge of equipment, drill sharpeners, blacksmiths and carpenters, and surface laborers. At an active mine the mechanical, drill-sharpening, and carpenter work usually can be handled by the regular mine staff.

Some shaft-sinking jobs entail starting from surface and involve clearing and preparing the site and installing a shaft collar and headframe, whereas others are merely deepening operations. In either instance the routine at the shaft bottom is the same, though dumping and other top arrangements will be somewhat different.

Assuming that the shaft is to be sunk from the surface, the first essential is a substantial shaft collar adequate to support the shaft timbers hung below it until they are blocked or held by shaft-bearers below, to prevent surface water from running into the shaft or draining into it by seepage through overburden above bedrock, and in some instances to support the front legs of the headframe. A well-designed concrete collar usually will fulfill all requirements, and to provide room for it the shaft may be sunk to bedrock of a size such that the inside dimensions of the concrete correspond with the outside measurements of the regular shaft sets. The concrete collar should be carried into bedrock far enough to insure a solid footing and to seal off surface water. In flat terrane, the collar usually is carried some distance above the ground, thus giving dump room around the shaft and providing for surface drainage away from the collar in all directions.

In ordinary shaft sinking in rock formations, the shaft is deepened by drilling rounds of holes in accordance with a standard arrangement. The holes are loaded and blasted in a sequence designed to obtain maximum breakage with a given amount of explosive. Thus, cut-holes are employed as in driving horizontal headings, though the procedure is somewhat different and varies with local conditions and the preferences of the operator. In heading work, it is generally desirable to break the rock as coarse as possible but small enough so it can be handled conveniently by one man or by the loading machine. In shaft work, however, it is best to break the rock as small as economically possible because of the greater ease in digging it out of the bottom of the shaft with pick and shovel, since a mucking sheet cannot be employed to shovel from as in driving headings.

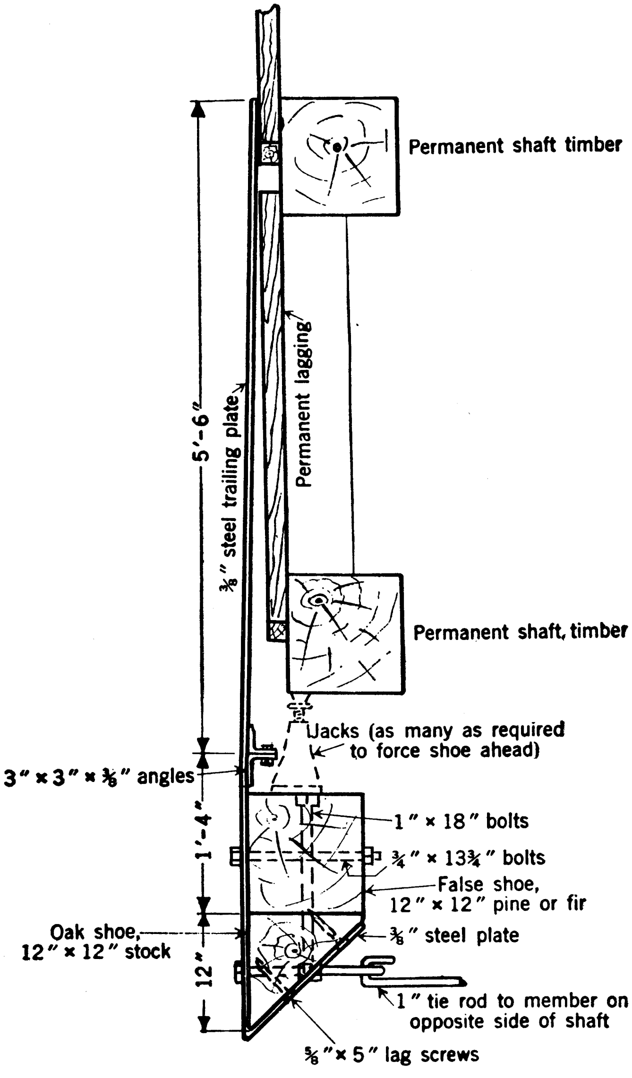
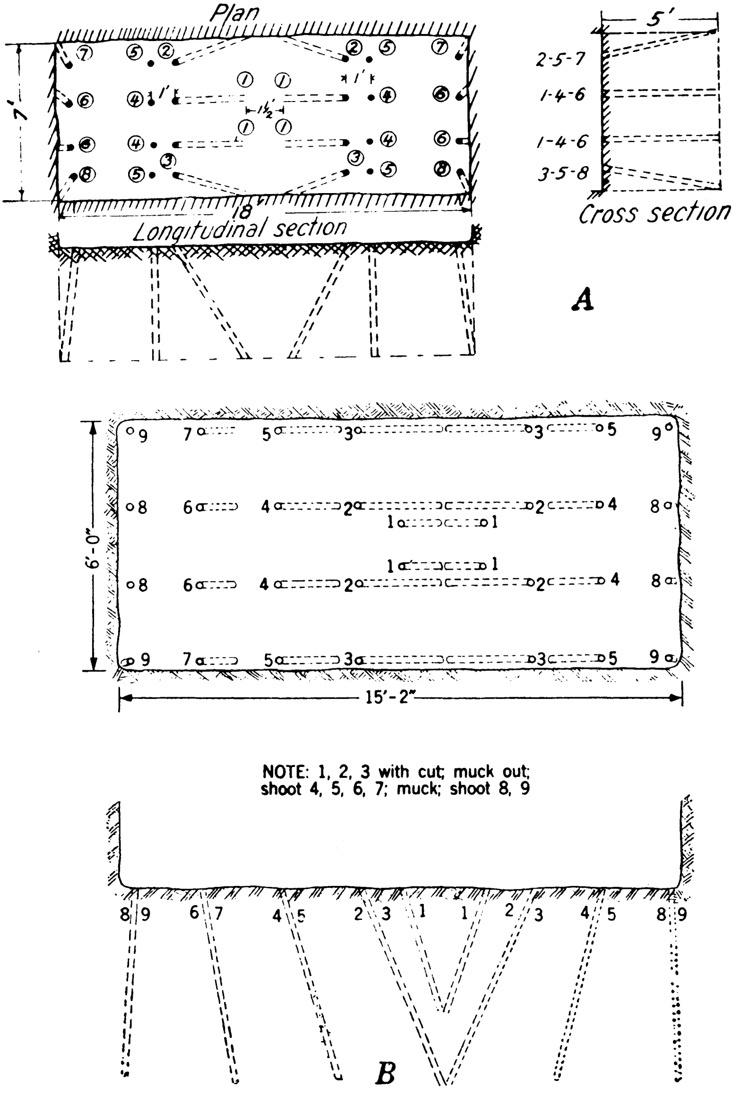

Permanent shaft timber
Permanent lagging
⅜" steel trailing plate
5'-6"
Permanent shaft timber
Jacks (as many as required to force shoe ahead)
3" × 3" × ⅜" angles
1'-4"
1" × 18" bolts
¾" × 13¾" bolts
False shoe, 12" × 12" pine or fir
Oak shoe, 12" × 12" stock
⅜" steel plate
12"
1" tie rod to member on opposite side of shaft
⅝" × 5" lag screws

FIGURE 44.—Shaft-sinking shoe; section through one side.

The cuts are commonly drilled in the form of a V (fig. 45) across the shaft if it is rectangular in cross section, or in the form of a diamond cut (fig. 46, *A* and *B*) if the shaft is square or circular, though the particular type of cut and details will depend upon the way the ground breaks and the size of the shaft. As in heading work, the relief and square-up holes are drilled and blasted to break in succession to the free faces provided by the cut holes and holes next preceding in the order of firing. Figure 46, *D*, shows a combination pyramid and V cut, in which the pyramid cut is shallow and reduces the burden on the first four holes of the deeper V cut. Figure 46, *C*, shows a benching round similar in principle to the slabbing round described under



NOTE: 1, 2, 3 with cut; muck out; shoot 4, 5, 6, 7; muck; shoot 8, 9

FIGURE 45.—Typical V-cut shaft rounds. *A*, Single-V cut. *B*, Double-V cut.

Driving Adits. With this round, half of the shaft area is drilled and blasted at one time, first on one end and then on the other.

The cut-holes in shaft work may also be termed "sump holes;" by blasting them and cleaning out the muck before blasting, and in some instances before drilling the rest of the round, they provide a sump for collection of water. The round shown in figure 46, *D*, performs a similar function. Cut-holes in shaft sinking may thus serve a dual purpose.

The standard V-cut may be drilled across the center or at one end, depending upon details of practice. By making the cut under the hoisting compartment and mucking it out first, the bucket may be lowered into it, making mucking from the benches less arduous. Sometimes a V-cut is drilled across one end of the shaft, blasted, and mucked out, and then the rest of the round is blasted progressively in a series of benches retreating toward the other end (fig. 45, *B*).

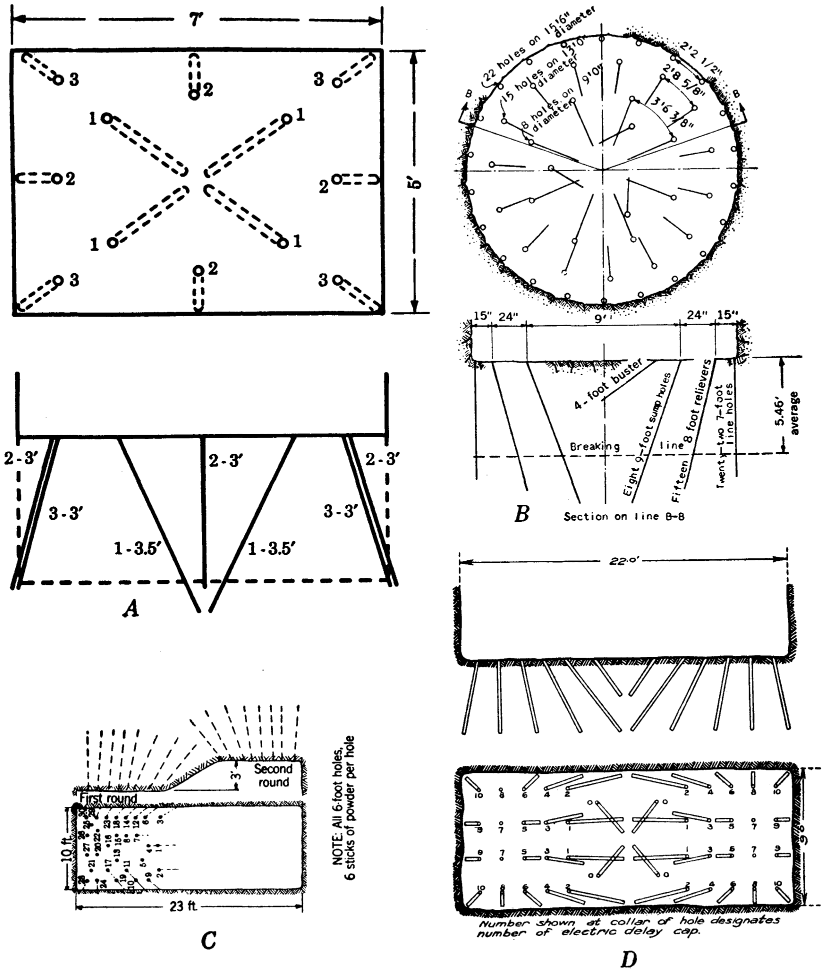

7′
5′
1
2
3
2-3′
3-3′
1-3.5′
A
22 holes on 15′6″ diameter
15 holes on 13′0″ diameter
8 holes on 9′0″ diameter
2′2 1/2″
2′8 5/8″
3′6 3/8″
B
15″
24″
9′
4-foot buster
Eight 9-foot sump holes
Fifteen 8 foot relievers
Twenty-two 7-foot line holes
Breaking line
5.46′ average
Section on line B-B
First round
Second round
3′
10 ft.
23 ft.
NOTE: All 6-foot holes, 6 sticks of powder per hole
C
22.0′
9′
Number shown at collar of hole designates number of electric delay cap.
D

FIGURE 46.—Types of shaft rounds. *A*, Pyramid cut in 5- by 7-foot shaft. *B*, Pyramid cut in circular shaft. *C*, Benching rounds. *D*, Combination of pyramid and V cut.

As stated under Shaft-Sinking Equipment, electric blasting is advocated for shaft work, and the holes are wired in series when fired from a blasting battery and in parallel when a power circuit is used. Large rounds of 60 to 80 holes are best fired from a power circuit with connections in series parallel. In series wiring, one leg wire of a detonator is connected to a leg wire of the second one, the other leg of which is connected, in turn, to the next one, and so on around until all holes are connected. This leaves two free ends, which are later connected to the lead wires after the circuit has been tested with a galvanometer and found not to be grounded. In parallel connection, two bus wires are laid along the shaft, one leg wire of each detonator being connected to one bus and the other leg to the opposite bus. Tests for grounding are made, and the round is then ready for connection to the lead wires. The latter are not connected to the source of power until everyone is out of the shaft and everything is in readiness for the blast. In battery firing, this source of power is the blasting machine, and the final connection is made at the binding posts of the machine. In firing from a power circuit, the final connection is made through a switch or, better, two switches, the first of which is open, the second being the firing switch. The latter should be in a locked box, the key to which is in the possession of the foreman or shift leader and which is so constructed that the switch cannot be closed unless the door is open, and then only by manual operation.

Mucking is usually the most time-consuming operation in conventional sinking in rock, and in some instances may take more time than all other operations combined. Thus, in sinking the No. 3 Frood shaft, which was 28 by 16 feet in cross section, 55.52 percent of the time was consumed in mucking operations.[89]

At the 20- by 6-foot Pim shaft,[90] 30.4 percent of the time was employed in mucking. In sinking the 9- by 25-foot Guadalupe shaft,[91] 50.9 percent of the time was spent in mucking, 24.6 percent in drilling and blasting, and 24.5 percent in timbering.

In sinking the McIntyre No. 11 shaft,[92] which was 17 by 24 feet in cross section, 49.2 percent of the time was spent in mucking, 15.5 percent in timbering, 11.3 percent in blasting and clearing gas from the shaft, 9 percent in drilling, 8.5 percent in scaling, 2.7 percent in hitch cutting, and 3.8 percent was consumed in delays. Twelve to 14 machines were used to drill about 66 holes aggregating 770 feet of hole per round in a double or triple V-cut arrangement. The rounds were blasted in three steps; first, the short cut holes and short easers, which were then mucked out, and then the long cut and bench holes, and last, light end holes and two or three center sump holes, which were drilled while the bench was being cleaned.[93]

In sinking the four-compartment No. 5 Magma shaft at the Magma mine,[94] 56.3 percent of the time was employed in mucking, 6.2 percent in blasting and clearing smoke, 25.0 percent in drilling, and 12.5 percent in timbering.

At the Argonaut mine, sinking a 7- by 17-foot shaft required one shift for drilling and blasting, three shifts for shoveling, and one shift for placing a set of timbers.[95]

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[89] Brock, Allan F., Sinking No. 3 Shaft at Frood: Eng. and Min. Jour., vol. 130, No. 9, Nov. 10, 1930, pp. 443–444.

[90] Poston, Roy H., Sinking Practice and Costs at the Pim Shaft, St. Louis Smelting & Refining Works, National Lead Co., St. Francois, Mo.: Bureau of Mines Inf. Circ. 6588, 1932, 13 pp.

[91] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares, Sonora, Mexico: Bureau of Mines Inf. Circ. 6307, 1930, 34 pp.

[92] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, 19 pp.

[93] Kee, H. A., Sinking Operation at McIntyre No. 11 Shaft, Porcupine District, Northern Ontario: Bull. Canadian Inst. Min. and Met., April 1928, 18 pp.

[94] Snow, Fred W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Bureau of Mines Inf. Circ. 6168, 1929, 32 pp.

[95] Vanderburg, W. O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Bureau of Mines Inf. Circ 6311, 1930, 14 pp.

Table 14 gives man-hour distribution at a number of typical shaft-sinking jobs.

TABLE 14.—*Man-hour distribution on typical shaft-sinking jobs*

Name	Size of shaft, feet	Drill round	Average rate of advance	Man-hours per foot of advance					
				Drilling	Blasting and blowing smoke	Mucking and hoisting	Timbering and concreting	Miscellaneous and delays	Total
Pim, Mo.[1]	6 by 20	Center V-cut, 32 holes, 10 drills.	3.94 feet per day	7.7	5.7	12.5	11.0	4.3	41.2
Kirkland Lake Gold, Ontario.[2]	6.33 by 14.33	Center V-cut, 50 holes, 2 machines on cross bars.	1.0 foot per 8-hour shift	12.7	Included with drilling.	45.7	11.2	14.2	83.8
Eureka-Asteroid, Mich.[3]	21 by 14	Center V-cut, 50 holes blasted in 2 stages.	7 feet in 41 hours	16.25	do	37.43	8.04		61.72
Caretta, W. Va.,[4] skip shaft	12 by 28	Triple center V-cut, 56 holes, 8 machines.	6 feet per round, 42.8 hours	10.5	6.5	42.7	5.4 timbering, 17.4 concreting.	22.8	105.3
Caretta, W. Va.,[4] air shaft	Circular, 19-foot diameter.	Triple-diamond cut, 50 holes, 8 machines.	6 feet per round, 48 hours	11.7	6.7	44.1	3.4 timbering, 18.3 concreting.	21.8	106.0
Morton Salt, Tex.[5]	Circular								
A. Caisson and seal section.	16-foot diameter		115 feet in 808 hours			39.8	Concreting 50.2	91.5	181.5
B. Rock section	do	Diamond cut, 26 holes	4.28 feet per round, 111 feet in 714 hours.	11.7	Included with drilling.	40.6	Concreting 32.0	71.1	155.4
C. Salt section	do	48 holes	5.46 feet per round, 451 feet in 6,816 hours.	7.4	do	10.7	Concreting 12.3	21.5	51.9
Matahambre, Cuba[6]	8 by 25	Center V-cut, 32 holes, 6 machines.	4.92 feet per round 2,057 feet in 17 months	[7] 10.26 [8] 4.18 [9] 14.44	do	[7] 27.60 [8] 6.70 [9] 34.30	[7] 12.06 [8] 8.38 [9] 20.43	[7] 13.58 [8] 5.76 [9] 19.34	[7] 63.50 [8] 25.02 [9] 88.52
Macassa, Ontario[10]	9 by 17	Center V-cut, 38 holes, 4 machines.	6.04 feet per 24 hours, 4.93 per round, 170 feet per month.	[11] 8.52	do	[11] 11.53	[11] 2.61	[11] 1.58	[11] 24.24

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[1] Poston, R. H., Sinking Practice and Costs at the Pim Shaft, St. Louis Smelting & Refining Works, National Lead Co., St. Francois, Mo.: Bureau of Mines Inf. Circ. 6588, 1932, 13 pp.

[2] Dumbrille, J. C., Mining Methods of Kirkland Lake Gold-Mining Co., at Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6490, 1931, 12 pp.

[3] Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.: Bureau of Mines Inf. Circ. 6348, 1930, 13 pp.

[4] Kelley, L. E., Shaft-Sinking Methods, Practices, and Costs at the Consolidation Coal Co. at its No. 216 Mine, Caretta, McDowell County, W. Va.: Bureau of Mines Inf. Circ. 6602, 1932, 15 pp.

[5] Taylor, M., Shaft Sinking at the Morton Salt Co. Mine at Grand Saline, Tex.: Bureau of Mines Inf. Circ. 6640, 1932, 8 pp.

[6] Homer, Dudley D., Sinking and Equipment of the No. 2 Shaft at Minas de Matahambre: Explosives Eng., vol. 11, No. 2, February 1933, pp. 38–62.

[7] Direct.

[8] Indirect.

[9] Total.

[10] Howes, G. A., and Jackson, Chas. F., Shaft-Sinking Methods and Costs of Plant and Equipment at Macassa Mine, Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6674, 1932, 10 pp.

[11] Data for 1 typical month and 171 feet sunk .

As already indicated, the entire round may be blasted and mucked out in one stage or, more commonly, the cuts are blasted and mucked out first, and the remaining holes are then blasted and mucked out in one or more additional stages.

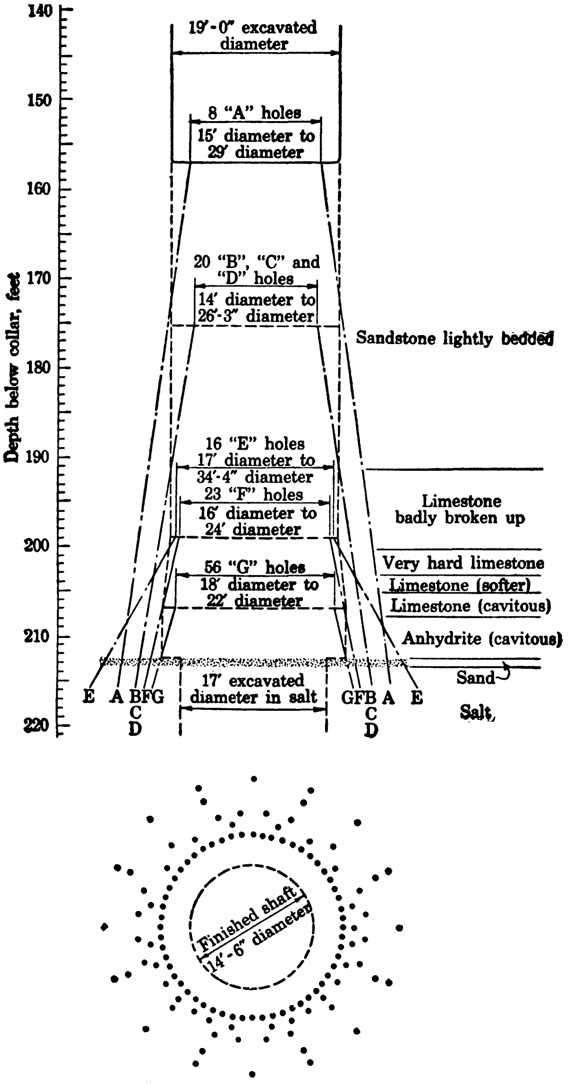

140
150
160
170
180
190
200
210
220
Depth below collar, feet
19'-0" excavated diameter
8 "A" holes
15' diameter to 29' diameter
20 "B", "C" and "D" holes
14' diameter to 26'-3" diameter
Sandstone lightly bedded
16 "E" holes
17' diameter to 34'-4" diameter
23 "F" holes
16' diameter to 24' diameter
Limestone badly broken up
56 "G" holes
18' diameter to 22' diameter
Very hard limestone
Limestone (softer)
Limestone (cavitous)
Anhydrite (cavitous)
Sand
Salt
17' excavated diameter in salt
E A BFG C D
GFB A E C D
Finished shaft 14'-6" diameter
PLAN OF GROUT HOLES AT SAND

FIGURE 47.—Grout holes drilled for introducing cement to seal off water.

From data already presented it is apparent that speed of sinking will depend largely on the speed with which the muck can be loaded and hoisted. The amount of water in the shaft also has an important influence upon the speed of sinking, and where considerable quantities must be handled progress is greatly retarded. Drilling water and small seepages may be bailed with the sinking buckets. Not only do raising and lowering pumps and extending water lines consume considerable time, but water retards drilling and mucking operations. If the inflow occurs only in a restricted section or horizon in the shaft, the problem is not so serious, because, as previously stated, it can be caught and kept from interfering with work at the shaft bottom or may be sealed off by grouting. If water comes in through fissured ground along considerable lengths of the shaft, it may be necessary to resort to grouting at numerous points, which involves frequent stopping of sinking, drilling of grout holes (fig. 47), rigging up for and pumping grout into the holes, and then waiting for the grout to set and for accumulated water to be pumped out. Conditions must be met as they arise, and a routine procedure ordinarily cannot be followed.

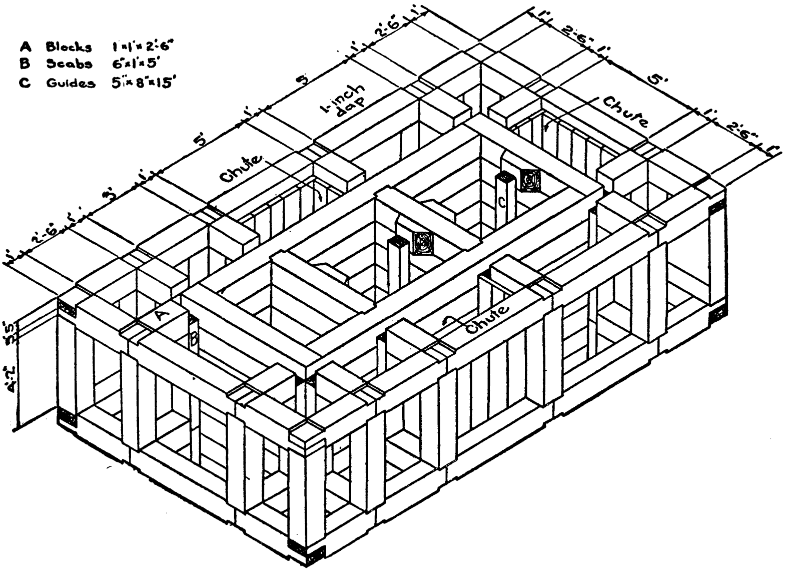


FIGURE 48.—Framing of jacket sets with cribbed shaft.

In good ground, timbering need not appreciably reduce the rate of progress, since it may be conducted concurrently with other operations from a suspended bulkhead or blasting set. In loose, sloughing, or swelling ground timbering must be kept close to the bottom and a set will usually have to be installed after each round is removed, which obviously adds to the time of the sinking cycle. If the ground is bad enough to require bridge or jacket sets (fig. 48), cribbed sets, or spiling, progress is necessarily slow, due not only to the additional timber that must be installed but to the care that must be exercised in removing the loose ground without causing runs. This may require the use of breast boards and bracing to hold back the ground while the sets are placed.

Shaft sets may be of timber, steel, or precast concrete beams. Timber is employed most commonly, although in recent years steel sets have been coming into favor owing to the availability of standard structural shapes suitable for this purpose and to reduction of fire hazards by their use. Precast concrete beams have not found general favor largely because of their weight. Timber sets have certain advantages in heavy ground since squeezing and crushing give warning of development of dangerous pressures, usually quite a while before actual failure occurs, and they can be removed by sawing and chopping with ordinary hand tools.

In recent years the practice of lining the entire shaft with concrete has grown. Not only does this provide a fireproof shaft, but if the shaft is to be used for ventilation purposes, the sides will be smooth and offer minimum resistance to the passage of the air current. In some instances concrete walls are poured after sinking has been completed, beginning at the bottom and removing the timbers installed during sinking as the concrete advances. In other instances concreting is done in sections of 50 to 100 feet or more as the shaft is deepened.

Timber sets are framed on the surface; and the separate members are lowered, one at a time, and hung in approximate position from the bearers or from the sets above by means of hangar bolts. When the set has been assembled in this manner, it is leveled and alined with plumb bobs, blocking and the threaded hanger bolts being used to hold it in place, and it is finally wedged tightly against the walls to prevent movement.

At intervals of 50 to 100 feet or more bearing sets are installed to hold any weight not sustained by the blocking, which may work loose owing to disintegration of the wall rock. These bearers also facilitate shaft-repair work that may be required later and that may involve removal and replacement of the original sets. The bearers consist of long, heavy beams placed across the shaft under the end plates of a regular shaft set and sometimes also under the dividers. Their ends rest in hitches cut into the wall rock.

Figure 49 shows methods of framing and assembling timber sets. Framing details vary somewhat at different mines, but those shown are typical. When framed as shown in figure 49, *A*, the wall plates are hung first, the end plates are then lowered and placed on top of the wall plates, corner posts or studdles are set, and usually after the set has been blocked and wedged at the corners the dividers and center posts are put in.

Figure 50 shows details of a steel shaft set. H-beams are well adapted to shaft sets because they give a wide bearing against the ground and at the same time provide a convenient method of supporting the lagging.

In firm ground that does not slack and slough on exposure to the air it may be unnecessary to lag the sets. In such instances the chief function of the timbers is to support the shaft guides, pipe lines, electric cables, and sheathing separating manway and hoisting compartments. Some ground will stand well without lagging if the air can be kept from it. This may be accomplished by guniting the walls with a coating of cement and sand.

When lagging is required, it may be of plank or of concrete slabs. Plank is lighter and therefore easier to handle and install and throws less weight on the sets and bearers. However, concrete slabs with steel sets provide a fireproof shaft.

For concreting shafts, various kinds of forms and methods of pouring are employed. Forms may be of wood or steel, or partly of wood and partly of steel, and may be built-in or of the removable

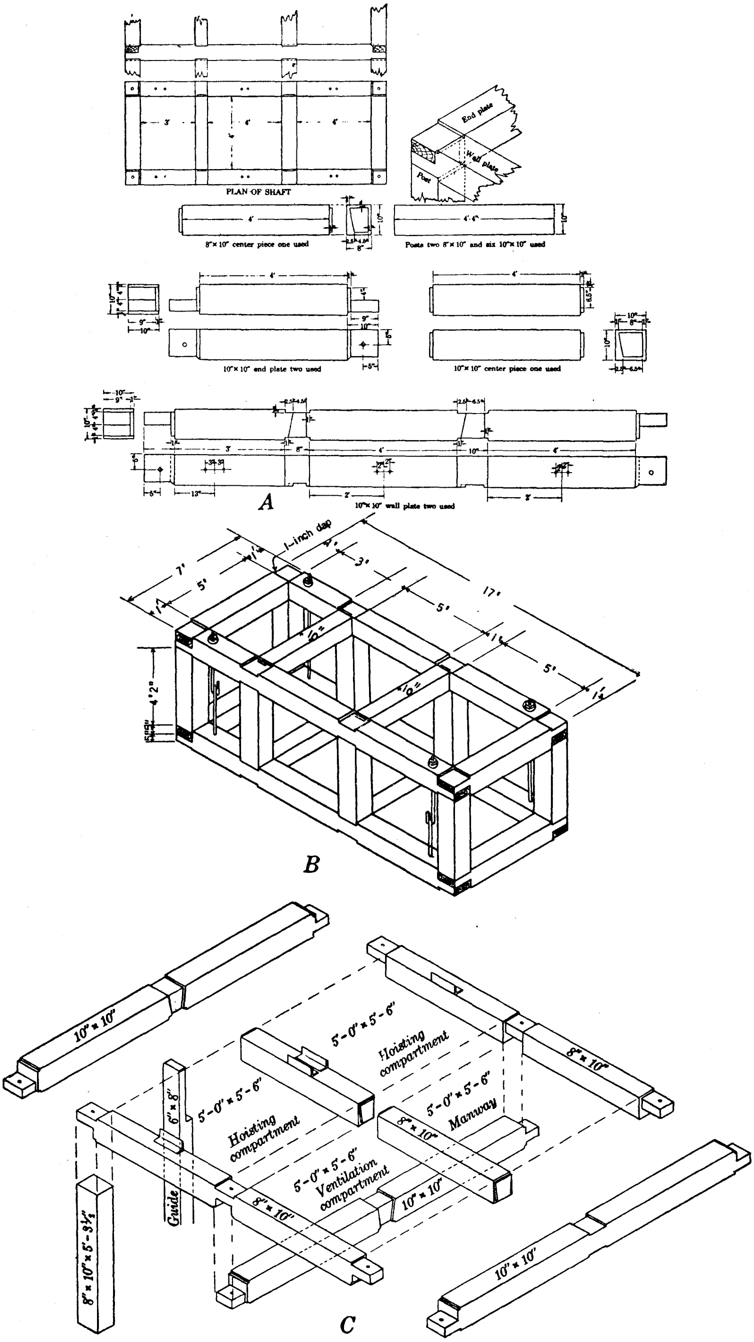

PLAN OF SHAFT
End plate
Wall plate
Post
8"× 10" center piece one used
Posts two 8"× 10" and six 10"× 10" used
10"× 10" end plate two used
10"× 10" center piece one used
10"× 10" wall plate two used
A
1-inch dap
7'
5'
17'
4'2"
B
10"× 10"
5'-0"× 5'-6"
Hoisting compartment
6"× 8"
Guide
8"× 10"× 5'-3½"
8"× 10"
Ventilation compartment
Manway
C

FIGURE 49.—Methods of framing and assembling timber shaft sets. *A*, Typical set for 3-compartment shaft *B*, Assembly of two sets. *C*, Typical timber set for 4-compartment square shaft.

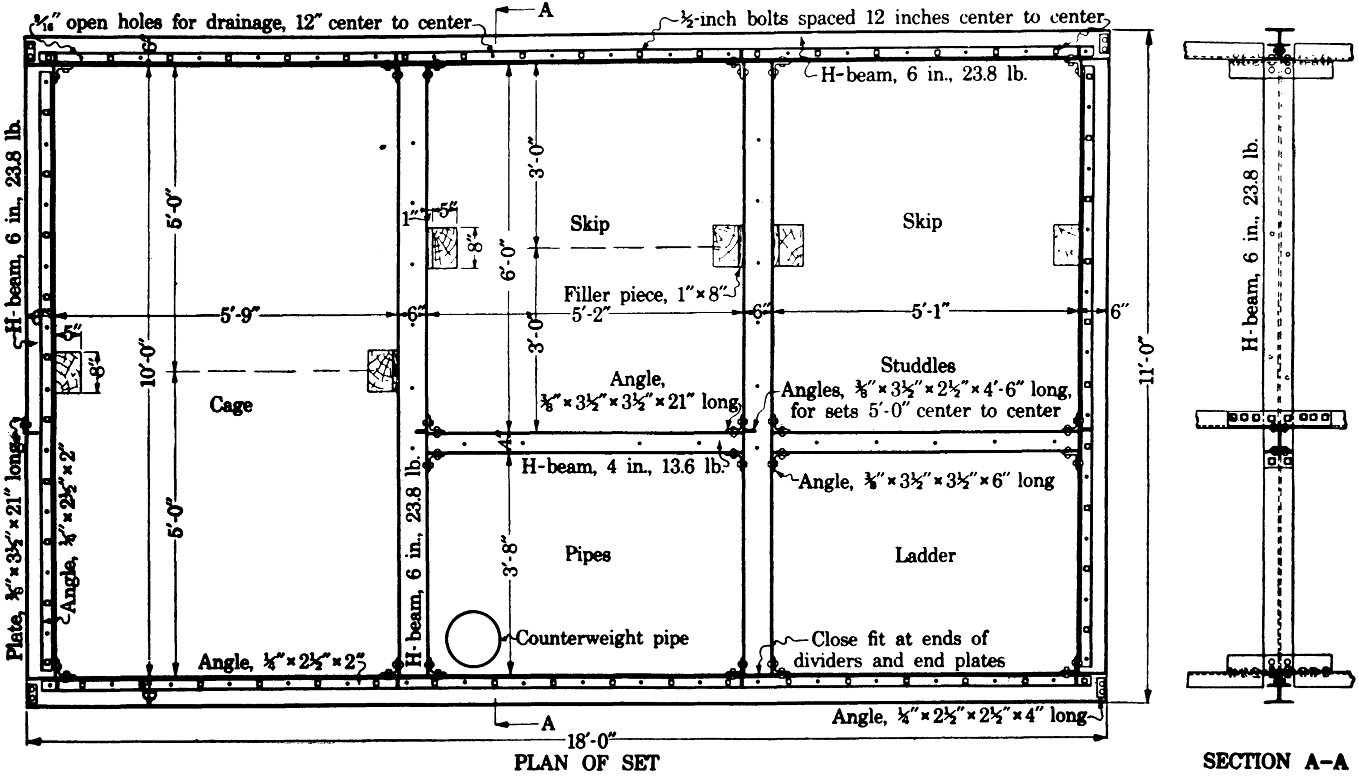
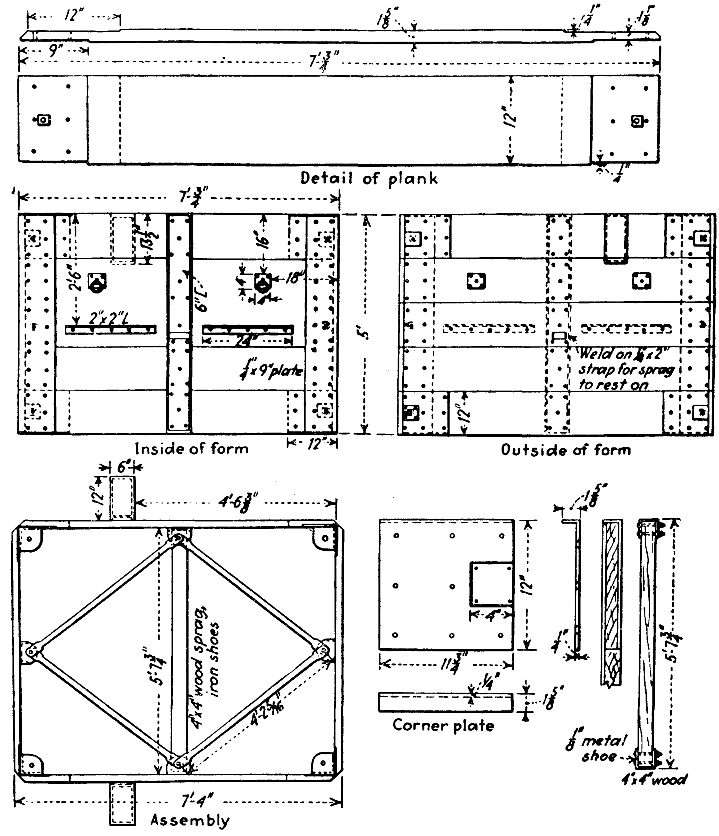
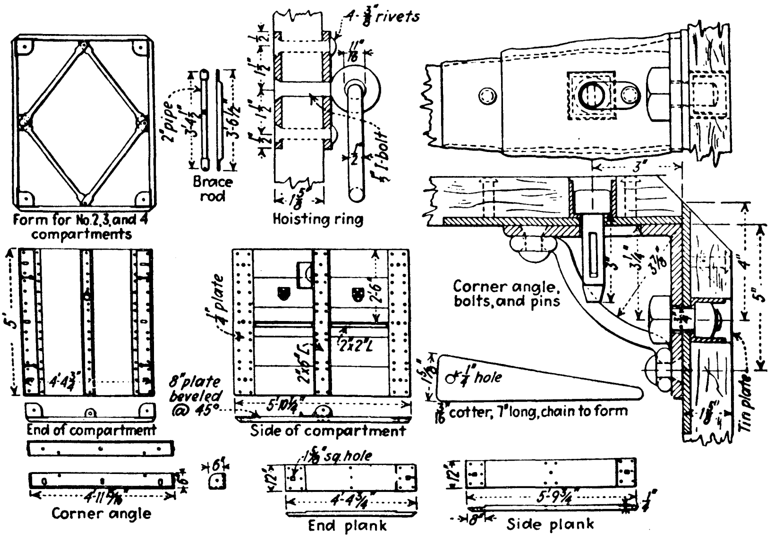

5/16" open holes for drainage, 12" center to center
A
½-inch bolts spaced 12 inches center to center
H-beam, 6 in., 23.8 lb.
H-beam, 6 in., 23.8 lb.
Plate, ⅜" × 3½" × 21" long
Angle, ¼" × 2½" × 2"
5'-0"
10'-0"
5'-0"
5'-9"
5"
8"
Cage
Angle, ¼" × 2½" × 2"
H-beam, 6 in., 23.8 lb.
1"
5"
8"
6"
3'-0"
6'-0"
3'-0"
4"
3'-8"
Skip
Filler piece, 1" × 8"
5'-2"
Angle, ⅜" × 3½" × 3½" × 21" long
H-beam, 4 in., 13.6 lb.
Pipes
Counterweight pipe
6"
5'-1"
6"
Skip
Studdles
Angles, ⅜" × 3½" × 2½" × 4'-6" long, for sets 5'-0" center to center
Angle, ⅜" × 3½" × 3½" × 6" long
Ladder
Close fit at ends of dividers and end plates
11'-0"
A
18'-0"
Angle, ¼" × 2½" × 2½" × 4" long
PLAN OF SET
H-beam, 6 in., 23.8 lb.
SECTION A–A

FIGURE 50.—Details of steel shaft set.


12"
9"
1 5/8"
1/4"
1 5/8"
7'-3/4"
12"
1/4"
Detail of plank
7'-3/4"
13 1/2"
16"
2'-6"
18"
2"x2"L
6"L
24"
1/4"x9" plate
5'
Weld on 1/4"x2" strap for sprag to rest on
12"
Inside of form
12"
Outside of form
6"
12"
4'-6 3/8"
5'-7 3/4"
4"x4" wood sprag, iron shoes
4'-2 5/16"
7'-4"
Assembly
12"
4"
11 3/4"
1/4"
1 5/8"
Corner plate
1 5/8"
1/4"
5'-7 3/4"
1/8" metal shoe
4"x4" wood

—Forms for combined dinky and pipe compartments

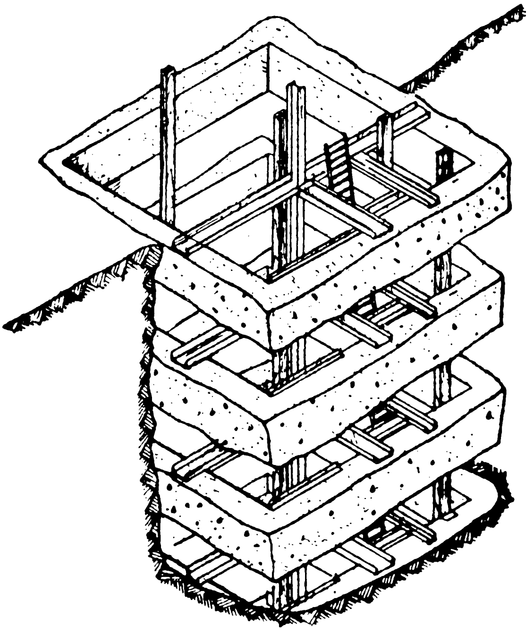

4-3/8" rivets
11/16"
2" pipe
3'-4 1/2"
3'-6 1/2"
Brace rod
1 1/2"
1 1/2"
1/2"
2"
3/4" 1-bolt
1 5/8"
Hoisting ring
Form for No.2,3,and 4 compartments
3"
Corner angle, bolts, and pins
3"
3 1/4"
3 1/8"
4"
5"
Tin plate
1 5/8"
5'
4'-4 3/4"
End of compartment
8" plate beveled @ 45°
1/4" plate
2'-6"
2"x2"L
2"x6"L
5'-10 1/4"
Side of compartment
1 5/8"
1/4" hole
3/16" cotter, 7" long, chain to form
4'-11 5/16"
6"
Corner angle
6"
1 5/8" sq. hole
12"
4'-4 3/4"
End plank
12"
5'-9 1/4"
1/4"
1/8"
Side plank

-General details of shaft forms

FIGURE 51.—Forms used in concreting the Denn shaft, Arizona.

sectional type. Built-in forms are of plank nailed to regular shaft sets used for sinking or to false sets of smaller outside dimensions, the regular sets being removed as the concrete walls are raised. After the concrete has set properly the plank is stripped off, and the false sets are removed. Removable forms of wood or steel are made in sections of a size convenient for handling, placing, and lowering through the shaft compartments. Figure 51 shows a form employed for concreting the Denn shaft in Arizona.[96] As a rule, only inside forms are used, the rock walls of the shaft furnishing the back support.

Thin concrete walls should be reinforced with steel, but walls thicker than about 10 or 12 inches usually will not require reinforcement. Concrete mixes are generally 1:2:4, 1:2:5, or 1:3:5, the latter probably being the one most commonly employed. The concrete is mixed on the surface and may be lowered in the shaft by means of buckets, in a car on a cage, or through a pipe. Concrete has been delivered successfully through pipe to depths of 3,000 feet or more. For great depths the pipe delivers first to a bucket provided with a spout, which in turn delivers behind the forms.



Sectional elevation - showing three rings in place

FIGURE 52.—Method of supporting shaft with concrete rings.

Figure 52 shows a method of supporting a shaft with concrete rings employed at the United Verde Mine.[97]

In lagged or concreted shafts, care should be taken to fill all open spaces behind the timber or concrete to prevent breakage by large slabs sloughing into the opening.

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[96] Brunel, F. P., Concreting the Denn Shaft: Eng. and Min. Jour., vol. 133, No. 12, December 1932, pp. 614–617.

[97] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

SPECIAL METHODS

The foregoing discussion refers to the conventional methods of sinking in rock formations. Caisson and freezing methods have been referred to already, and it was pointed out that when such methods are employed contracts for the work usually are let to specialists who have the required equipment and are experienced in such work.

New shafts at operating mines are often raised instead of sunk. Raising eliminates the expense of mucking and pumping and the delays incident to hoisting the muck through the shaft. In raising, the shaft usually is carried from one level of the mine to the next in lifts of 100 to 200 or, in rare instances, 300 to 600 feet. It may be raised full size; or, more commonly, a small pilot raise is put up first, which is then enlarged from the top down to full size of the shaft, timbers being installed as in conventional sinking operations, the muck being drawn off through the bottom.

One objection to raising is the difficulty of ventilating, which is enhanced as the raise becomes longer. To overcome this objection it has been found feasible to put down a drill hole first along the axis of the proposed shaft and then raise on the drill hole. By placing an injector-type ventilator in the drill hole at the top of the raise or an ejector-type ventilator at the top of the drill hole, good ventilation can be obtained. The ejector type is preferable as it is outside the raise, so that the compressed-air line can be connected to it and the raise thus cleared of smoke before the miners re-enter after blasting. In very long raises the handling of tools and supplies likewise becomes laborious and time-consuming, and workmen expend an undue amount of time and energy in getting to and from the working place. Hence, shorter lifts usually are more economical.

A shaft may be carried up full size in the form of a small shrinkage stope, as shown in figure 53.[98] In this instance a cribbed manway was carried up in each of two corners of the shaft, and the miners worked on top of broken rock supported on a timber bulkhead at the level as shown. Blasting was done in three stages—*A*, *B*, and *C*. After each part of the round had been blasted, entrance to the face was gained through one of the cribbed manways by pushing back the inclined 8-foot timbers. The particular job illustrated would have been greatly accelerated and the cost lessened considerably had the shaft been raised on a borehole with a compressed-air ejector at the surface or had a fan been installed at the level for blowing up one manway and down the other. As it was, the cost was only about half the estimated cost of sinking. The broken rock supported the shaft walls during raising, and timber was put in after the raise broke through to surface by drawing off the rock from below to make room for successive sets from the top downward.

A recent innovation in shaft-sinking methods, for which wider use in the future is predicted, was drilling with a shot drill the full diameter of the shaft.[99] A 5-foot circular shaft was drilled to a depth of 1,125 feet through gabbro, serpentine, and ankerite by means of a shot drill close-connected to a motor-driven driving mechanism installed in a cage or cab (fig. 54), the whole being lowered to the bottom of the shaft on cables wound on the drum of a hoist. The drill consists of a circular steel core barrel 15 feet long with a removable cutting shoe 1½ inches thick by 12 inches high attached to the lower end. This drill is rotated at about 60 r. p. m. on chilled-steel shot, which grinds an annular ring around a central core of rock. A tapered ring with a number of fluted dogs around the inside and at the bottom of the core lifter served to grip, break off, and raise the core in lengths up to 8 or 10 feet.

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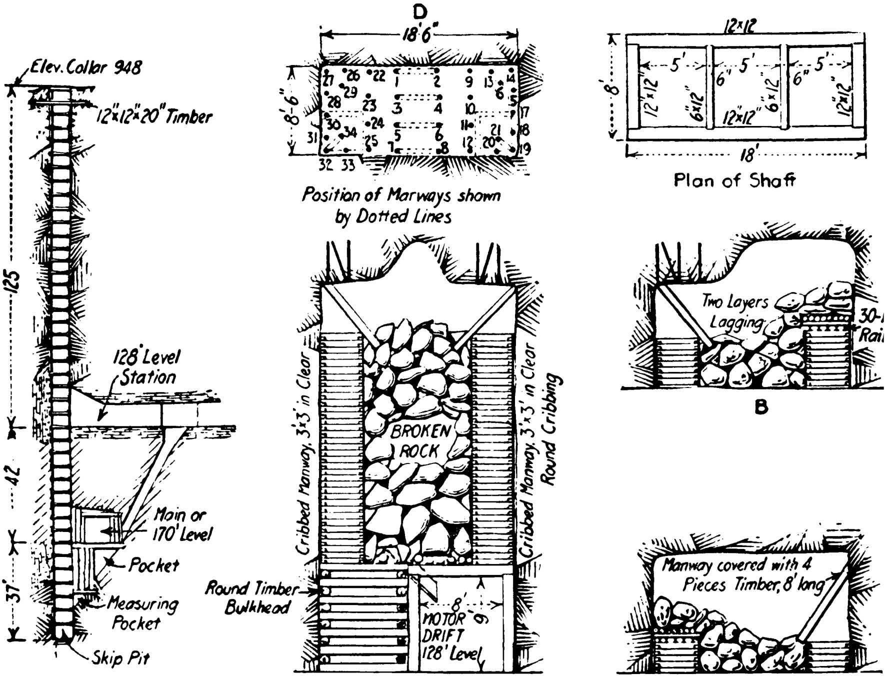
[98] Jackson, Chas. F., Shaft Raising at the Harold Mine: Eng. and Min. Jour., Apr. 7, 1917, p. 625.

[99] Newson, J. B., and Jackson, C. F., Shaft Sinking with a Shot Drill, Idaho-Maryland Mine, Grass Valley, Calif.: Bureau of Mines Inf. Circ. 6923, 1936, 10 pp.


D
18' 6"
8' 6"
Position of Manways shown by Dotted Lines
Elev. Collar 948
12"x12"x20" Timber
125
128' Level Station
42
Main or 170' Level
37
Pocket
Measuring Pocket
Skip Pit
12x12
5'
6"
6x12
12"x12"
8'
18'
Plan of Shaft
Cribbed Manway, 3'x3' in Clear
BROKEN ROCK
Round Cribbing
Round Timber Bulkhead
8'
MOTOR DRIFT 128' Level
Two Layers Lagging
30-
Rai
B
Manway covered with 4 Pieces Timber, 8' long
A
C

SECTIONS SHOWING THE HAROLD SHAFT AND DETAILS IN THE OPERATION OF RAISING

A shows condition after blasting cut holes B shows condition after blastin west end. C shows condition at end of round D shows plan of round

FIGURE 53.—Method of raising shaft full size.

Although the operation was an experimental one and mechanical and other difficulties had to be overcome, suprisingly low costs were obtained. The actual cost of the drilling operation was $23.57 per foot, whereas the equipment, including head frame, transformers, and hoist, cost $15,588 or $13.86 per foot. Most of this equipment, of course, could be used again on another job. Three 60-inch holes drilled close together would be at least the equivalent in capacity of a three- or four-compartment shaft. By drilling the shaft, the walls were not shattered and weakened by blasting as in ordinary shaft-sinking operations, and only short sections required artificial support, which was obtained by concreting. A three-deck cage that would accommodate 10 men to a deck comfortably was installed later in the shaft.

As this is written, another shaft, 66 inches in diameter, has been sunk by this method to a depth of 694 feet in greenstone; the ultimate depth is to be 1,200 feet.[1]

SHAFT-SINKING COSTS

Costs of shaft sinking vary with the size and depth of the shaft; nature of the rock (the ease with which it can be drilled and broken, and the nature and amount of support required); amount of water that must be handled, speed of sinking, organization of the work, and the equipment employed; and labor and other local conditions.

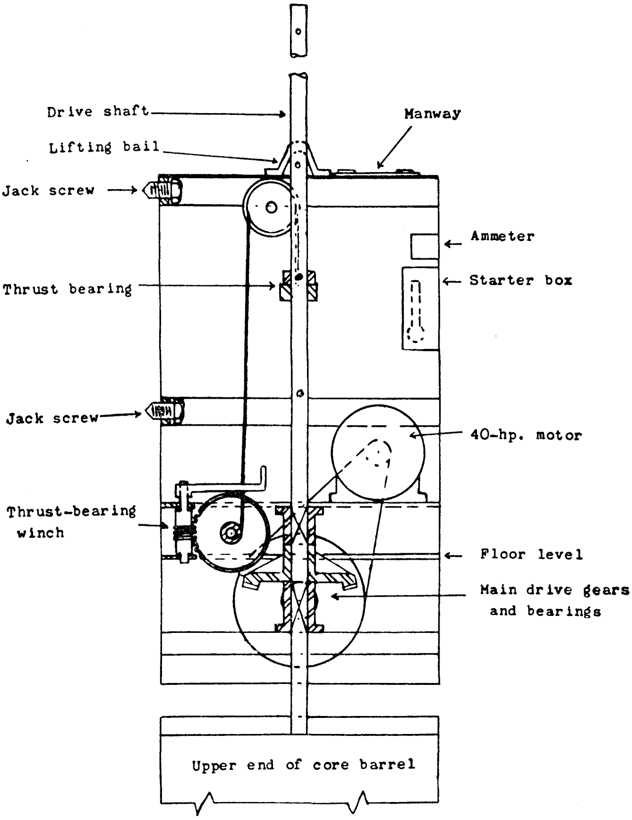

Drive shaft
Lifting bail
Manway
Jack screw
Ammeter
Starter box
Thrust bearing
Jack screw
40-hp. motor
Thrust-bearing winch
Floor level
Main drive gears and bearings
Upper end of core barrel

FIGURE 54.—Generalized cross section of shaft drill.

The effect of most of these factors upon costs is self-evident, but effect of the rate of sinking is not so apparent. There is probably a certain rate of sinking that will vary somewhat with each job, which will result in the lowest cost per foot of shaft and below or above which this cost will increase.

---

[1] Eng. and Min. Jour., New Advance Scored in Boring Holes of Large Diameter: Vol. 139, No. 10, October 1938, pp. 39-41. Information obtained after the shaft had been completed is that the final cost, exclusive of equipment, was: Labor, $12.16; power and supplies, $7.34; total, $19.50 per foot; and that total time required was 7 months.

If the exigencies of the case require the greatest possible speed in sinking, certain economies otherwise possible cannot be achieved. Speed work requires a high degree of organization and a routine, which, if temporarily interrupted, may disrupt the operations for some time; extra labor not normally required may be employed as an insurance against interruptions, which obviously adds to the cost. Furthermore, crowding the bottom of a shaft with a large crew does not make for economy.

A striking example of economy by slow-speed sinking is afforded by the experience at the Burra-Burra mine in Tennessee,[2] where the three-compartment McPherson shaft was sunk 6 feet 7 inches by 16 feet inside the timbers. By sinking at a leisurely rate averaging a little over 2 feet per day, a saving of $50 per foot was made as compared to previous higher-speed sinking. During a typical 25-day working month when an advance of 57 feet was made, costs per foot were as follows:

Labor cost (including bonus)	$41. 02
Explosives	5. 80
Material other than explosives	14. 63
Air-drill expense and power	4. 09
Total cost per foot	65. 54

No time was lost from powder sickness or bad air, since 8 hours elapsed between blasting and resumption of work in the bottom of the shaft. In earlier work higher costs were attributable to (*a*) large force of men in shaft, (*b*) unpleasant working conditions, (*c*) greater hazards, and (*d*) periodic upsets in organization.

Table 15 presents some typical shaft-sinking costs.

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[2] Weaver, Lamar, Shaft Sinking in Tennessee: Eng. and Min. Jour., vol. 129, No. 6, Mar. 24, 1930, pp. 301-302.

TABLE 15.—*Typical shaft-sinking costs*

Name, location, and year	Size of shaft, feet	Kind of rock and water conditions	Depth sunk, feet	Shaft support	Cost of shaft					
						Sinking costs per foot sunk				
					Preparation and equipment	Labor and supervision	Explosives	Supports, timber and concrete	Other supplies, power, miscellaneous	Total sinking
No. 2 Tri-State district,[1] 1921.	5 by 7, 1 compartment.	Surface, shale, dolomite, flint beds.	260	Cribbing in surface and shale, 100 ft.; balance untimbered.						$20.08
Pim, Southeast Missouri,[2] prior to 1931.	6 by 18 in clear, 3 compartments.	Clay, limestone, shale. 60 to 70 g. p. m. of water.	730	Concrete collar. Timber to hold guides.	$2,645.00	$43.06	$4.83	$4.67	$23.61	76.17
Pecos, N. Mex.,[3] 1929	6.67 by 20.33 outside of timbers; 3 compartments.	Metamorphosed granite and diorite (schist).	219 from 831 to 1,050	Timber sets		68.71	([4])	([4])	([4])	110.25
Capote, Cananea, Mexico,[5] 1925.	28 by 8.5; 5 compartments.	Quartz porphyry (?); some water.	212 from 1,400- to 1,600-ft. levels.	do	2,074.23	61.40	5.11	22.08	24.55	113.14
Pilares, Mexico,[6] 1924, 1925.	12 by 20 rock dimensions.	Monzonite porphyry, shaft sunk on pilot raise.	379.5 from 1,400 to 1,800 levels.	do		[7] 2.85 [8] 19.49	[7] 1.73 [8] 1.14	[7] .30 [8] 29.07	[7] 2.19 [8] 11.02	[7] 7.07 [8] 60.72
No. 6 United Verde,[9] Ariz.	14.5 by 14.5	Quartz porphyry raised full size from level to level.	2,500 from 500 to 3,000	Concrete rings (fig. 51*B*).		[10] 52.07 [11] 17.65	[10] 8.47 	 [11] 26.51	[10] 16.57 [11] 5.95	[10] 77.11 [11] 50.11
South shaft, Teck-Hughes, Ontario,[12] 16 months, 1929–1930.	12.5 by 13	Syenite and lamprophyre.	2,284	Timber sets		70.38	7.55	14.50	28.32	120.75
Jim Crow; 85 mine,[13] N. Mex., completed 1930.	7.75 by 16.5 rock dimensions.	Andesite and diorite	1,720	Timber sets kept close to bottom.		34.55	2.84	8.55	10.68	56.62
No. 5 Magma [14] mine, Ariz.; prior to 1929.	8 by 21 rock section, 4 compartments.	Firm sedimentary and slabby diabase. 5 gal. water per minute.	2,531	Timber sets	30,220.91	71.26	4.12	14.76	14.36	104.50
Edith, Ariz., 1921 [15]	7 by 16 rock section, 3 compartments.		575 ft. and 2 stations, 80 days.	Concrete		[16] 20.70	([17])	([17])	([17])	[16] 43.14
Do.[15]	do		603 ft. and 5 stations, 69 days.	do		[18] 11.31	([19])	([19])	([19])	[18] 31.35

See footnotes at end of table.

TABLE 15.—*Typical shaft-sinking costs*—Continued

Name, location, and year	Size of shaft, feet	Kind of rock and water conditions	Depth sunk, feet	Shaft support	Cost of shaft					
					Preparation and equipment	Sinking costs per foot sunk				
						Labor and supervision	Explosives	Supports, timber and concrete	Other supplies, power, miscellaneous	Total sinking
Macassa, Ontario, 1931, 1932.[20]	9 by 17, rock section, 3 compartments.	Conglomerate, 1,449 ft. syenite, porphyry, lamprophyre; 1,040 ft. heavy flow water one section grouted.	2,489	Timber sets	$66, 228. 00	$36. 18	$6. 37	$6. 30	$22. 08	$70. 93
Central Patricia,[21] Ontario, 1930.	6.5 by 16 ft. outside the timbers; 3 compartments.	Greenstone. Heavy flows of water at 250 and 280 ft.	527	do	[22]55, 000. 00	[23] 23. 56	7. 55	9. 47	37. 78	78. 36
Ashley,[24] Ontario, 1931	6.2 by 17 outside timbers, 3 compartments.	Basalt. Considerable water.	436 inclined 53°	do		[23] 44. 19	5. 46	13. 15	[25] 56. 29	119. 09
San Antonio,[26] Manitoba, 1927.	6 by 17; 3 compartments	Some water	556	do		30. 26	5. 38	5. 06	27. 80	68. 50
U. S. mine,[27] Bingham, Utah, prior to 1936.	6.33 by 15.5 outside timbers, 3 compartments.		422 from 2,000 to 2,400 levels.	do		46. 06	3. 11	26. 51		75. 68
Saginaw;[28] Bisbee, Ariz., 1930.	9 by 16 rock section, 3 compartments.	Limestone; no water	837 from 915 to 1,752	do		39. 04	8. 63	8. 14	7. 56	63. 37
Hecla, Idaho,[28] 1927–1929.	9 by 21.5, rock section, 4 compartments.	Quartzite; 10 gal. water per minute.	829 from 2,212 to 3,042	do		61. 69	8. 23	9. 60	18. 21	97. 73
Morning, Idaho,[28] 1924	9 by 26, rock section, 4 compartments.	Quartzite	200 from 2,450 to 2,650	do		80. 44	6. 46	12. 85	15. 72	115. 47
Sylvanite,[29] Ontario, 1931.	6 by 16, rock sections, 3 compartments.	Syenite and syenite porphyry.	963 from 1,000 to 1,963	do		34. 13		9. 56	27. 40	71. 09
Vipond,[29] Ontario, 1931	6 by 16, 3 compartments.	Schist; considerable water.	252 from 1,200 to 1,450 levels.	do	3, 270. 27	35. 85	6. 99	8. 78	[30] 29. 11	80. 73
Matahambre,[31] Cuba	8 by 25, rock section, 4 compartments.	Quartzite and shale. Not over 5 gal. per minute of water.	2,057	Steel sets; 81 ft. concreted, 12 sets of bearers.	37, 256. 00	66. 02	6. 70	45. 90	46. 92	165. 54

McPherson,[32] Tenn.	8 by 17.33 outside timbers; 3 compartments.	Schist and graywacke	57 ft., 1 mo.	Timber sets		41.02	5.80	18.72		65.54
Caretta,[33] W. Va.:										
A. Skip shaft	12 by 28 rock	Sandstone and shale. Considerable water at 300 ft. Considerable grouting required. (Fig. 47.).	563	Concrete		106.20		[34] 53.65	50.74	210.59
B. Manway shaft	do		572			130.55		[34] 62.73	67.81	261.09
C. Air shaft	19 ft. diam. circular		562			82.15		[34] 25.73	48.49	156.37
Pioneer,[35] British Columbia.	7 by 14 rock section	Greenstone and diorite	From 1,700 to 3,200, 7 ft. per 24 hrs.	Timber sets		29.93	5.07	3.50	12.18	50.68
Do.[35]	do	do	Raised, 625 ft. from 1,700 level.	do		10.40	5.15	4.50	6.28	26.33
Falconbridge,[36] Ontario	[37]22.33 by 16.2	Sand, gravel, boulders, wet quicksand.	125	Steel sets, concrete	50.55	115.05	2.60	111.45	[38] 2.35	282.00

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[1] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.: Bureau of Mines Inf. Circ. 6121, 1929, 11 pp.

[2] Poston, R. H., Sinking Practice and Costs at the Pim Shaft, St. Louis Smelting and Refining Works, National Lead Co., St. Francois, Mo.; Bureau of Mines Inf. Circ. 6588, 1932, 13 pp.

[3] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[4] All supplies, $41.54.

[5] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Bureau of Mines Inf. Circ. 6247, 1930, 37 pp.

[6] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares, Sonora, Mexico: Bureau of Mines Inf. Circ. 6307, 1930, 34 pp.

[7] Pilot.

[8] Shaft.

[9] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[10] Excavation.

[11] Concreting.

[12] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6322, 1930, 21 pp.

[13] Youtz, R. B., Mining Methods at the Eighty-Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.: Bureau of Mines Inf. Circ. 6413, 1931, 26 pp.

[14] Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Bureau of Mines Inf. Circ. 6168, 1929, 32 pp.

[15] D'Arcy, R. L., Mining Practices and Methods at the United Verde Extension Mining Co., Jerome, Ariz.: Bureau of Mines Inf. Circ. 6250, 1930, 11 pp.

[16] Concreting cost only, and including 2 stations.

[17] All supplies, $22.44. Concreting cost only, and including 2 stations.

[18] Concreting cost only, and including 5 stations.

[19] All supplies, $20.04. Concreting cost only, and including 5 stations.

[20] Howes, G. A., and Jackson, Chas. F., Shaft-Sinking Methods and Costs of Plant and Equipment at Macassa Mine, Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6674, 1932, 10 pp.

[21] Keast, A. J., and Jackson, Chas. F., Method and Cost of Exploring, Equipping for Development, and Developing the Central Patricia Group of Claims, Northern Ontario: Bureau of Mines Inf. Circ. 6681, 1933, 13 pp.

[22] Approximate.

[23] Shaftmen only. Other labor included under "Supports" and "Other supplies."

[24] Emens, W. H., and Jackson, Chas. F., Methods and Costs of Developing and Equipping the Ashley Gold Mine, Matchewan Gold District, Ontario: Inf. Circ. 6707, 1933, 28 pp.

[25] Includes first cost of rock drills, drill steel, rails, boiler fuel, gasoline, etc.

[26] Jackson, Chas. F., Some Notes on Methods and Costs of Equipping and Developing Prospects: Bureau of Mines Inf. Circ. 6693, 1933, 24 pp.

[27] Walker, L. A., Shaft Sinking in an Operating Mine: Min. and Met., April 1937, pp. 202–204.

[28] Gardner, E. D., and Johnson, J. F., Shaft-Sinking Practices and Costs: Bureau of Mines Bull. 357, 1932, 104 pp.

[29] Jackson, Chas. F., and Knaebel, J. B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, 151 pp.

[30] Includes $11.69 hoisting cost.

[31] Homer, Dudley D., Sinking and Equipment of the No. 2 Shaft at Minas de Matahambre: Explosives Eng., vol. 11, No. 2, February 1933, pp. 38–62.

[32] Weaver, Lamar, Shaft Sinking in Tennessee: Eng. and Min. Jour., vol. 129, No. 6, Mar. 24, 1930, pp. 301–302.

[33] Kelley, L. E., Shaft-Sinking Methods, Practices, and Costs of the Consolidated Coal Co. at Its No. 261 Mine, Caretta, McDowell County, W. Va.: Bureau of Mines Inf. Circ. 6602, 1932, 15 pp.

[34] Steel and concrete.

[35] Fawley, Allan P., Mining Gold Ore at Pioneer: Canadian Min. and Met. Bull. 305, September 1937, Transactions Sec., pp. 475–486.

[36] Oliver, R. M., Shaft Sinking Through Heavy Overburden at Falconbridge: Canadian Min. and Met. Bull. 311, March 1938, pp. 103–111.

[37] Sinking shoe 26.25 by 19.9.

[38] Power not included.

LATERAL DEVELOPMENT

In tabular, dipping lodes and thick massive deposits the ore bodies are developed laterally at several horizons spaced at more or less regular vertical intervals. In areas of high relief each horizon may be developed by driving an adit into the side of the hill, from which branch headings are driven as required.

When the main extraction opening is a shaft, the horizons or "levels" are developed from stations cut into the side of the shaft. Shaft stations usually are partly cut and timbered during shaft-sinking operations to avoid later damage to the permanent shaft timbers from blasting against them when level development is begun. Special shaft timbering is required at the stations, and by installing it at the time the shaft is being sunk, the work of tearing out regular shaft sets and replacing them with station sets later on is obviated. By cutting the stations during shaft sinking, level development may be begun on one or several levels at a time with minimum delay as soon as the shaft has been completed.

A level comprises lateral workings (stations, drifts, and crosscuts) at approximately the same horizon or elevation, though the term "level" is often employed loosely to designate all workings tributary to the level proper. Level development provides haulageways and means of access to the working places. The main drifts and crosscuts serve as a base from which to drive auxiliary drifts, crosscuts, raises, and winzes for the purpose of exploring and blocking out the ore bodies and preparing them for economical extraction by stoping. They also serve as airways for ventilation of the workings, afford drainage facilities, and carry compressed-air and water lines and electric wiring for operation of mining equipment.

LEVEL INTERVAL

The vertical distance between main levels is commonly 100, 125, or 150 feet, although intervals as low as 50 feet and as high as 250 or 300 feet sometimes are employed. In many instances arbitrary intervals of 100 to 150 feet are satisfactory, but in others the most suitable and economical interval can be determined only by careful weighing of a number of factors governed by the shape and dip of the lode, position of the shaft in relation to the ore bodies, distribution of and distance between separate ore bodies, persistence of the ore between levels, and physical characteristics of the ore and wall rocks, which, in turn, determine the mining method to be employed.

These and other factors are discussed in an earlier Bureau of Mines publication.[3] In this publication the author has produced a set of curves showing the relationship between cost per ton of ore mined and level interval for shrinkage stoping, horizontal cut-and-fill stoping, and square-set-and-fill stoping based upon estimates of development costs and of the number of tons of ore developed per vertical foot for different level intervals under a series of assumed conditions. These curves indicate broadly that for these methods of mining the cost per ton of ore mined decreases rapidly from intervals of 60 feet to intervals of 100 to 140 or 150 feet, but very slowly, if at all, for greater intervals.

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[3] Vanderburg, William O., Factors Governing the Selection of the Proper Level Interval in Underground Mines: Bureau of Mines Inf. Circ. 6613, 1932, 17 pp.

It is obvious that, other things being equal, the greater the amount of ore developed per foot of development headings driven, the lower will be the cost of development per ton of ore mined. Were no other factors to be considered, it might be concluded that the greater the level interval, the less the development and, hence, the lower the cost of development per ton of ore.

Within variable limits this is broadly true, but other factors enter in to establish these limits. Thus, if the interval is too great, difficulties in ventilation may be experienced; the cost of driving connecting raises between levels usually increases rapidly beyond lengths of 100 to 150 feet; in irregular, erratic deposits a long level interval usually will not serve to explore the ore bodies adequately; or it may require so long a time to mine out the ore between two levels that costs of level maintenance will be excessive and the stope walls may become so heavy that stopes cannot be kept open long enough to permit extraction of all the ore.

A short level interval has the following advantages: (1) More thorough exploration is possible, which is especially important in irregular, erratic deposits where the ore is not continuous; (2) shorter connecting raises are required between levels; (3) handling of steel, explosives, and timber into the stopes is facilitated; (4) more points of attack by stoping can be made available; (5) in heavy ground requiring the use of slow extraction methods the time required to work out a stope is less than with a high interval, and since the weight on a stope or sloughing of the walls often increases considerably the longer it is kept open, expense of maintenance and loss of ore may be decreased considerably with a short interval; (6) good ventilation usually is maintained more easily.

Among the disadvantages of the shorter level interval are: (1) Greater development cost due to increased number of stations, greater footage of drifts and crosscuts with correspondingly more trackage and pipe lines; (2) more levels to maintain; (3) a greater proportion of the ore tied up in drift and floor pillars, which may have to be left permanently or extracted later at considerable extra expense.

The stoping method to be employed may have an important bearing on the level interval. Thus if the mining system requires grizzly levels above the haulage levels, or if floor pillars are required below the drifts, the effective stoping height will be reduced considerably; and if the level interval is short, the pillars will constitute a large proportion of the ore between levels.

Where block caving is the mining method employed haulage levels generally are located to come under or near the bottom of the ore body, and in modern practice high columns of ore up to 300 feet or more are undercut and caved in one lift. Opening of grizzly and undercutting levels above the main level is a part of stope development that will be discussed later.

In sublevel stoping a level interval of 200 feet or more usually is advantageous if the ore body is thick or deep enough.

If ground conditions require short stope lifts for rapid working out of the stopes before weight or sloughing becomes excessive, or if erratic ore occurrence demands a short level interval for prospecting purposes, it may be most economical to drive the haulageways in the footwall and either to crosscut to the lode at frequent intervals (connecting the crosscuts with extraction drifts in the ore) or to drive inclined raises in rock from the haulageway to the bottom of the stopes for ore extraction. Footwall drifts, of course, add to the cost of dead work but may materially reduce the cost of maintaining the haulageways.

In tabular deposits dipping at flat angles, a given vertical interval between levels obviously will provide a longer stope lift measured on the dip; and the flatter the dip, the greater the relative length of the stope lift. Such deposits usually are mined through inclined shafts, and the level interval is measured along the dip rather than vertically and determined by the height of stope desired or by the requirements for prospecting the ore bodies.

In flat-lying deposits occurring in the form of disconnected ore bodies at different horizons the level interval may be variable, and levels may be established at elevations to match these horizons.

Where the ore is in widely separated shoots along a vein or fault or in the form of irregular disconnected bodies separated by barren rock the dead work required to get from one ore body to another may be reduced considerably if conditions are such that the ore can be mined from sublevels not connected to the shaft and the ore transferred to one main haulage level.

If a dipping vein deposit is worked through a vertical shaft in the footwall, crosscuts from it to the lode become progressively longer on each successive level, and dead work on the lower levels will be excessive, especially if the level interval is short or the dip flat. The choice is then between a high level interval or an inclined shaft, although a vertical shaft starting in the hanging wall and passing through the lode midway between the top and bottom of the ore will reduce the dead work considerably.

SIZES OF DRIFTS AND CROSSCUTS

The size of headings will be influenced by such considerations as strength of ground and requirements as to timbering for its support; type of haulage to be employed, whether hand or animal tramming, storage-battery or trolley locomotives, and size of cars to be used; type of loading chutes or platforms; economical size from the standpoint of cost of driving; stoping method to be followed; ventilation requirements; and amount of water to be drained through ditches.

The size of main openings with respect to requirements for ventilation, haulage, and drainage has been discussed already in connection with the driving of adits. The minimum size to meet these requirements is generally desirable from the standpoint of cost of driving and maintenance. When drifting in ore that stands well, it may be more economical to drive drifts larger than the minimum, especially if mechanical mucking is employed, since the cost per foot will not increase in proportion to the increase in size, whereas the cost per ton of ore broken in the drifts will decrease with increased size of heading. At one large gold mine in Canada it was found that by driving wide drifts the cost per ton of ore from the drifts was only 1.8 times the cost per ton of stope ore.[4] In steeply dipping tabular deposits, development drifts in ore are often driven the full width of the ore if it is not too wide. If the ore is thick or the horizontal section of the ore body is wide by reason of its flat dip, the drift is often carried along the footwall and is the most economical size, first cost and maintenance cost considered.

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[4] Jackson, Chas. F., Shrinkage Stoping: Bureau of Mines Inf. Circ. 6293, 1930, p. 33.

The stoping method to be employed may influence the size of the drifts. Thus, if shrinkage or cut-and-fill stoping is to be done directly on the drift timbers, it may be desirable to drive high drifts rather than return later and take down backs preparatory to stoping. On the other hand, if pillars are to be left over the drifts and the stopes silled at a higher level, the drift would be kept low to take as little height from the stoping lift to the next level as possible.

Table 16 gives data on dimensions of drifts and crosscuts that are typical of American practice.

TABLE 16.—*Sizes of typical drifts and crosscuts*

Mine and location	Character of ore or rock		Rock dimensions, feet		Type of haulage	Remarks
	Drifts	Crosscuts	Drifts	Crosscuts		
Homestake, S. Dak.	Hornblende schist and quartz.	Hard schist and porphyry.	6.5 by 7 to 7 by 8	7 by 7.5 to 8 by 8	Compressed-air motors, 1-ton cars.	Usually untimbered.
Alaska-Juneau, Alaska	Slate and metagabbro	Slate and metagabbro	9 by 9	9 by 9	Trolley, 10-ton cars	Untimbered.
Argonaut, Calif.	Quartz veins in slate	Slate, greenstone, schist	9 by 9 to 11 by 10	9 by 9	Hand, 1-ton cars	Drifts heavily timbered.
Kennedy, Calif.	do	do	9 by 9	8 by 11	do	Do.
Empire, Calif.	Quartz veins in granodiorite.	Granodiorite	5 by 7	5 by 7	Hand and mule. Storage-battery locomotive on 4,600 level.	Untimbered.
Sixteen to One, Calif.	Quartz veins in amphibole schist.	Amphibole schist	6 by 7	6 by 7	Hand and animal	Do.
Hollinger, Ontario	Quartz and mineralized basaltic schist.	Basaltic schist and porphyry.	8.5 by 7	8.5 by 7 to 9 by 7.5.	Trolley and storage battery, 1½-and 3-ton cars.	Both timbered and untimbered.
McIntyre, Ontario	do	do	8 by 8	8 by 8	Trolley, 16, 20, and 40 cu. ft. cars.	Do.
Vipond, Ontario	do	do	6 by 7	6 by 7	Hand, 1-ton cars	Do.
Lake Shore, Ontario	Hard silicified porphyry and lamprophyre.	Hard porphyry, syenite, and lamprophyre.	8 by 8	8 by 10	Storage battery, 2-ton cars; (1930), 2-ton cars.	Drifts timbered.
Teck-Hughes, Ontario	do	do	5.5 by 7.5, later widened to 8.5.		Hand, 1-ton cars	Do.
No. 1, Tri-State district	Dolomite and chert	Dolomite	7 by 7	7 by 7	Hand, mules, trolley, 1-ton cars.	Untimbered.
No. 8, Southeast Missouri.	Dolomitic limestone	Dolomitic limestone	7 by 8	7 by 8	Trolley, 1- and 2-ton cars	Do.
Bonne Terre, Southeast Missouri.	do	do	7.5 by 8 to 9 by 10. Main haulage, 8 by 10 min.	7.5 by 8 to 9 by 10.	Trolley, 2½-ton cars	Do.
Edwards, N. Y.	Silicified dolomite	Dolomite and gneiss	6 by 7	6 by 7	Storage-battery locomotive, 1-ton cars.	Usually untimbered.
Mascot No. 2, Tenn.	Dolomitic limestone	Dolomitic limestone	8 by 8 (main haulage)	7 by 7	Trolley, 2-ton cars	Untimbered.
Bunker Hill, Idaho	Quartzite and ore	Quartzite	7 by 8	15 by 8	Storage-battery locomotive, 2- and 3-ton cars.	Drifts timbered.
Page, Idaho	do	do	8 by 9.5 (6 by 8 inside timbers).	8 by 9.5	Storage-battery locomotive, 2.3-ton cars.	Do.
Hecla, Idaho	do	do	8 by 8 in clear, min. In ore, 13 ft. high by full width of ore up to 18 ft.	5 by 7 in clear	Storage-battery locomotive, 5-ton cars.	Do.

Park-Utah, Utah	Quartzite, limestone, and ore.	Quartzite and limestone.	8 by 8	5 by 7	Trolley, 3½-ton cars	Drifts usually timbered.
Magma, Ariz	Altered silicified diabase or porphyry and quartz.	Mostly sedimentary rocks.	7 by 7 or 7 by 9 untimbered; 10 by 10 timbered.	7 by 7 to 7 by 9	Trolley and storage battery, 2-ton cars.	Drifts mostly timbered.
Engels, Calif	Sheared and altered diorite.	Diorite	6 by 7	6 by 7	Storage-battery locomotive, 1-ton cars.	Untimbered.
Marquette No. 1, Mich	Hard iron ore	Slate, quartzite, jasper, and altered diorite.	8 by 8	8 by 8	Trolley and storage-battery locomotives, 5.5-and 2.5-ton cars.	Do.
Marquette No. 2, Mich	Soft iron ore	Ferruginous cherts and slates.	10 by 10		Trolley, 5.5- and 2.5-ton cars.	Do.

Drifting and crosscutting practices have been discussed by Gardner [5] in an earlier Bureau of Mines publication.

In general, the same types of rounds, methods of drilling, blasting, mucking, and ground support are employed as those previously discussed under the caption Methods, Equipment, and Costs of Driving Adits.

COST OF DRIVING DRIFTS AND CROSSCUTS

Costs of lateral development vary between rather wide limits and depend on the size of the headings; nature of the ore and rock, its drillability, breakability, and ground support needed; rate of driving; type of equipment, whether hand or mechanical loading is employed; wage rates and efficiency of labor; and management. These points have been discussed already under Costs of Driving Adits. It has also been pointed out that driving costs will depend on the amount of overhead they must carry and thus that at a producing mine where such costs are distributed between production and development, the unit cost may be considerably lower than where development work only is being done.

Table 17 presents typical drifting and crosscutting costs.

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[5] Gardner, E. D., Drilling and Blasting in Metal Mine Drifts and Crosscuts: Bureau of Mines Bull. 311, 1929, 170 pp.

TABLE 17.—*Typical drifting and crosscutting costs*

Mine and year	Size of rock section, feet	Kind of rock	Drill round	Mucking method	Rate of advance	Costs per foot						
						Labor and supervision	Compressed-air drills, steel	Explosives	Timber	Track	Miscellaneous	Total
Mineville, N. Y., 1927	15 by 10	Gneiss	28 holes, V-cut	Power shovel	5 ft. per shift	$9. 90	[1] $1.80	$2. 36	$0. 00	(2)	$1. 40	$15. 46
Tri-State No. 1, 1928	7 by 7	Limestone and chert	12 holes	Hand	6 ft. per round	7. 37	1. 38	2. 38	. 00	$0. 46	2. 54	14. 13
Southeast Missouri No. 8, 1928.	7 by 8	Limestone	16 holes, side draw cut	do	3 to 6 ft. per round	4. 89	1. 13		. 00		[3] 1. 85	7. 87
Mascot, Tenn., 1929	7 by 7	Dolomitic limestone	20 holes, pyramid cut	do	6.5 ft. per round	4. 07	1. 57	2. 25	. 00		. 28	8. 17
Hanover-Bessemer, N. Mex.	8 by 7	Limestone, granodiorite, iron ore.	Center cut	do		6. 13	2. 41	2. 05	. 83	. 44	. 18	12. 04
Teziutlan, Mexico, 1930–31.	6 by 8	Metamorphosed igneous and sedimentary.	17 holes	do		8. 83	2. 01	2. 50	. 82		3. 30	17. 46
Bunker Hill & Sullivan, Idaho, 1928.	7 by 8	Quartzite	14 holes, bottom draw cut.	do		5. 01	1. 17	2. 41	. 46		2. 38	11. 43
Questa, N. Mex., 1930	4.5 by 6.5	Porphyry	7 or 8 holes	do	4 ft. per round [4]	[4] 3.79 [5] 5.47	[4] 1. 93 (5)	[4] 1. 25 [5] . 70	. 14 . 14	[4] . 30 [5] . 15		7. 41 6. 46
Morenci, Ariz., 1928	10 by 10 arched	Monzonite porphyry	22 holes	Power shovel	5.65 ft. per round	6. 76	1. 38	1. 42	1. 99		1. 88	13. 43
Ray, Ariz., 1928	8.5 by 9	Quartz, sericite schist, some porphyry.	13–15 holes, V-cut	Hand	5 ft. per round	7. 43	[6] 1. 54	. 95	4. 37		. 50	14. 79
Verde Central, Ariz., 1927.	5 by 7	Vein matter greenstone, quartz porphyry.		do	4 ft. per round	6. 19	2. 26	2. 71			. 54	11. 70
Central Patricia, Ontario, 1930.	5 by 7	Quartz vein		do		6. 21	4. 36	3. 50		. 37	. 34	[7] 14. 78
	5 by 7	Greenstone		do		6. 57	4. 52	3. 54	. 02	. 64	. 36	[7] 15. 65
Siscoe, Quebec, 1935	8 by 8	Greenstone and granodiorite.	20 to 24 holes, pyramid cut.	Power shovel	747.3 ft. in 1 month	8. 83	1. 18	2. 80		. 90	. 91	14. 62 [8] 1. 85 16. 47
Britannia, British Columbia, 1934.	10 by 12	From hard and blocky to soft and schistose.	29 to 33 holes	Scraper slide	6 ft. per shift							[9] 25. 83
Hollinger, Ontario, 1934.[10]	7 by 8.5 crosscuts.	Schist and porphyry	20 holes, pyramid cut	Hand	6 ft. per round	[11] 6. 47	1. 68	2. 35			. 27	[11] 10. 77
Do	Drifts	Quartz veins in schist	do	do	do	[11] 8. 47	2. 25	2. 35			. 10	[11] 13. 17
Page, Idaho, 1936	5 by 7	Quartzite		Power shovel	5 ft. per round	5. 56	[12] . 87	2. 39	1. 76		2. 35	12. 93
Pioneer, British Columbia,[13] 1936, upper levels.	do	Greenstone, diorite, quartz.	15–19 holes, burnt cut	Hand	5.5 ft. per round	[11] 6. 00	2. 16	(14)				[11] 8. 16

See footnotes at end of table.

TABLE 17.—*Typical drifting and crosscutting costs*—Continued

Mine and year	Size of rock section, feet	Kind of rock	Drill round	Mucking method	Rate of advance	Costs per foot						
						Labor and supervision	Compressed-air drills, steel	Explosives	Timber	Track	Miscellaneous	Total
Pioneer, British Columbia,[13] 1936, lower levels.	do	do	do	Power shovel	6.0 ft. per round	[11] $6.20	$2.62	(15)			$0.13	[11] $8.95
Siscoe,[16] August 1935	8 by 8	Granodiorite	20–24 holes, pyramid cut.	do	743.3 ft. in 1 month (6-day week).	8.84	1.17	$2.80		$0.90	.91	[17] 14.62
Octave, 1935	6 or 7 to 10 by 7.	Granodiorite and quartz vein.	12 to 18 holes	Hand	3.5 to 4.0 ft. per round.	[18] 6.87	2.08	1.38	$0.07	.14	.45	10.99

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[1] Includes track.
[2] Included with compressed air, drills, and steel.
[3] Includes explosives.
[4] Machine drilling.
[5] Hand drilling.
[6] Machine drills only.
[7] Costs prior to production period.
[8] Hoisting and waste disposal.
[9] Includes proportion of capital charges, equipment, and construction expense.
[10] Young, Arthur W., Mining [Hollinger Consolidated Gold Mines, Ltd.]: Canadian Min. Jour., September 1935, pp. 377–379.
[11] Direct costs only.
[12] Power for drilling and mucking.
[13] Fawley, Allan P., Mining Gold Ore at Pioneer: Canadian Min. and Met. Bull. 305, September 1937, Transactions Sec., pp. 475–486.
[14] Included with labor, 14 lbs. per foot.
[15] Included with labor, 17 lbs. per foot.
[16] Brown, W. F., Driving a Long Crosscut: Canadian Min. Jour., November 1936, pp. 579–581.
[17] Not including hoisting and waste disposal, $1.845.
[18] Includes tramming, hoisting, and shop labor.

The figures are not necessarily directly comparable, even where they cover costs under similar conditions because of variations in the methods employed by different companies in distributing overhead charges, power costs, etc.

Table 18 gives typical performance, man-hour, and supply-consumption data on drifting and crosscutting.

TABLE 18.—*Typical performance, man-hour, and supply consumption, drifts and crosscuts*

Mine	Size, feet	Rock	Drill, round	Mucking method	Man-hours per foot				Explosives, pounds per foot	Timber, per foot	Power, kw.-hr. per foot
					Drilling and blasting	Mucking	Other	Total			
Spring Hill, Mont., 1930	7 by 7	Limestone and diorite	12 holes	Hand	3.03	3.03		6.06	9.40	None	46.0
Granada, Quebec	5 by 7	Graywacke, conglomerate, and vein.	18 holes	do	4.70	2.35	2.58	9.63	15.67	do	63.0
Teziutlan, Mexico	6 by 8	Metamorphosed sediments and igneous rocks.	17 holes	do	7.73	14.77	15.48	37.98	10.97	29.6 bd. ft	251.4
Pecos, N. Mex	7 by 8	Granitic schist and vein	12 to 16 holes	do				13.00	9.40	16.4 lin. ft	46.3
Morenci, Ariz	10 by 10	Granodiorite	22 holes	Power shovel				8.16	10.45	107.88 bd. ft	
Ray, Ariz	8.5 by 9	Quartz-sericite schist	13 to 15 holes	Hand				9.52	6.92	113.0 bd. ft	
Eureka-Asteroid, Mich., 1928.		Iron formation, chert, and quartzite.	24 holes	Scraper slide							
a. Untimbered	8 by 10	In hard rock			4.27	.82	.82	5.91	16.00	None	
b. Timbered	9 by 11	In soft rock			3.71	1.35	3.38	8.44	8.00	74.0 bd. ft	
Montreal, Wis	9 by 9	Chert, slate, greenstone	15 to 22 pyramid cut.	do	4.41	2.02	2.77	9.20	11.6	64.0 bd. ft	
Do	8 by 8	do	do	do	4.98	2.86		7.84	12.9	None	
Do	5 by 7	Iron ore		Hand	1.83	2.02		3.85	6.2	do	
Do	9 by 12	do	30 holes	Scraper slide	3.94	1.09	2.21	7.24	7.4	89.0 bd. ft	
Burra-Burra, Tenn., 1928	8 by 8	Graywacke, conglomerate, schist, massive sulfides.	20 holes	Hand	3.30	3.30		6.60	11.2	None	36.0
Marquette No. 2, Mich	10 by 10	Soft iron ore	21 holes, center V cut.	Power shovel	.83	2.00	1.91	4.74	25.5		
Champion, Mich	8 by 13.5	Trap	35 holes	Scraper slide	4.90	6.20	3.10	14.20	17.0	11.0 bd. ft	
Morning, Idaho, 1929	13 by 13	Vein material and quartzite	25 to 26 holes	Power shovel				28.80	20.0	149.1 bd. ft	
El Potosi, Mexico, 1933	5 by 7	Soft limestone	18 holes, toe cut	Hand				3.49	8.21	None	
Braden, Chile	7.5 by 9.5	Andesite porphyry	9 to 12 holes	do	1.73	3.27	[1] 3.20	8.20	5.50	137.8 bd. ft	
Butte and Superior, Mont., 1930.	7 by 9 and 8 by 10, timbered.	Granite and vein matter, mostly hard.	12 holes	do				9.60	12.5	45.0 bd. ft	
Page, Idaho, 1928	8.5 by 9	Soft, loose, swelling		do				4.80	15.7	79.0 bd. ft	

---

[1] Timbering

PLAN OF LEVEL LAY-OUT

The level lay-out will depend on the size, shape, dip, and distribution of the ore bodies, the strength of the ore and wall rocks, the mining method to be employed, the anticipated rate of production, and the type of haulage.

In the Tri-State and Southeast Missouri districts, wide, flat, bedded deposits of firm ore with strong capping are mined by breasting out the full width and height of the ore but leaving pillars for support of the back. The level plan thus becomes the ore outline. If the shaft is sunk outside the ore body, a short connecting crosscut from the shaft, and perhaps others to separate ore bodies, constitute all the development work required, the ore being mined as the workings advance. (Fig. 55.)

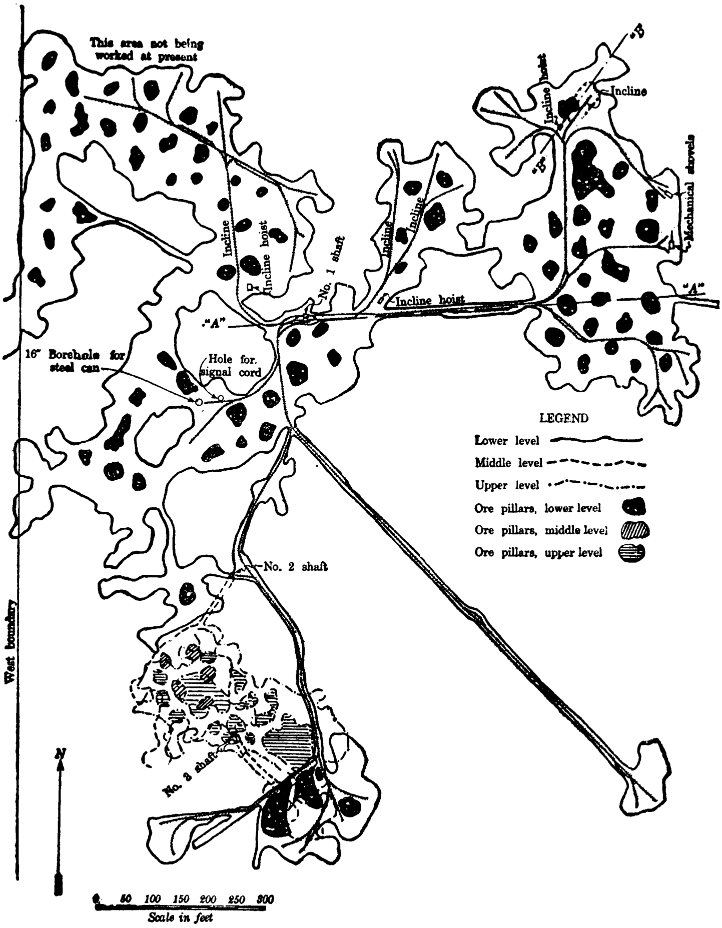
This area not being worked at present
Incline
Incline hoist
No. 1 shaft
"A"
16" Borehole for steel can
Hole for signal cord
Incline
Incline
Incline hoist
Incline hoist
Incline
"B"
"B"
Mechanical shovels
"A"
No. 2 shaft
No. 3 shaft
West boundary
N
LEGEND
Lower level
Middle level
Upper level
Ore pillars, lower level
Ore pillars, middle level
Ore pillars, upper level
0 50 100 150 200 250 300
Scale in feet

FIGURE 55.—Plan of Barr mine, Tri-State district.

Figure 56, *A*, shows a partial plan of a level at the Morning mine in Idaho, where the ore is in a vein 6 to 30 feet wide, dipping at 80° to nearly vertical. The deep levels are developed by a shaft in the footwall sunk from an adit level. Levels are established at 200-foot vertical intervals, and level development comprises the cutting of shaft stations and skip pockets, crosscuts to the vein with extensions to provide room for tail track, pump, and motor-charging stations near the shaft and drifts on the vein. Drifts are 13 by 13 feet in cross section and are timbered with standard drift sets.

Where the vein is much wider or flatter, as at the Bunker Hill and Sullivan mine, crosscuts are driven from the shaft or incline to and through the ore, but more than one drift may be required to facilitate rapid removal of broken ore from the stopes. (Fig. 56, *B* and *C*.) Here the dip is 40° to 50° and the footwall drift (fig. 56, *C*) is used as a main haulageway and base from which to drive rock raises to the upper part of the stopes. If the ore were the same width and vertical, a single extraction drift through the center might be employed; or, where very wide, two or more parallel drifts might be required.

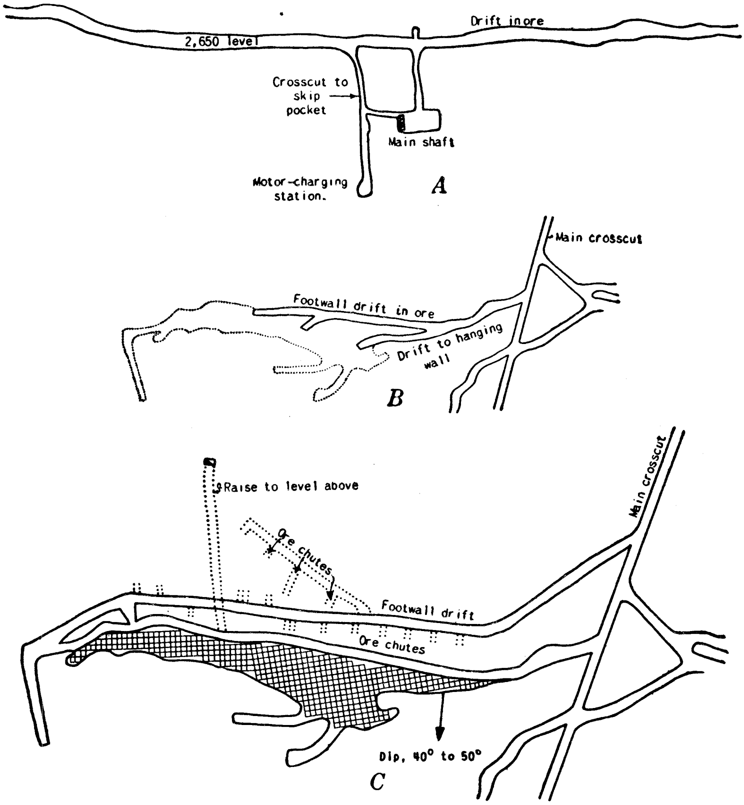
Drift in ore
2,650 level
Crosscut to skip pocket
Main shaft
Motor-charging station.
A
Main crosscut
Footwall drift in ore
Drift to hanging wall
B
Raise to level above
Ore chutes
Main crosscut
Footwall drift
Ore chutes
Dip, 40° to 50°
C

FIGURE 56.—*A*, Plan of level, Morning mine, Idaho. *B*, Partly developed level in wide ore body. *C*, Complete development of sill floor, Bunker Hill and Sullivan mine.

Figure 57, *A*, is a plan of the 550-foot level of the Hollinger mine in Ontario as it was in 1924 and where a network of parallel and cross lenses or veins of ore (shown in solid black) is developed from a main haulage drift in the footwall of the ore zone, from which crosscuts are driven approximately at right angles and about 300 feet apart to intersect the various veins. Extraction drifts are then driven in the veins from these crosscuts.

Figure 57, *B*, is part of a level plan at the Homestake mine (1930). The ore body is wide and the ore was mined in transverse shrinkage stopes between pillars, which were mined later by square-set stoping.

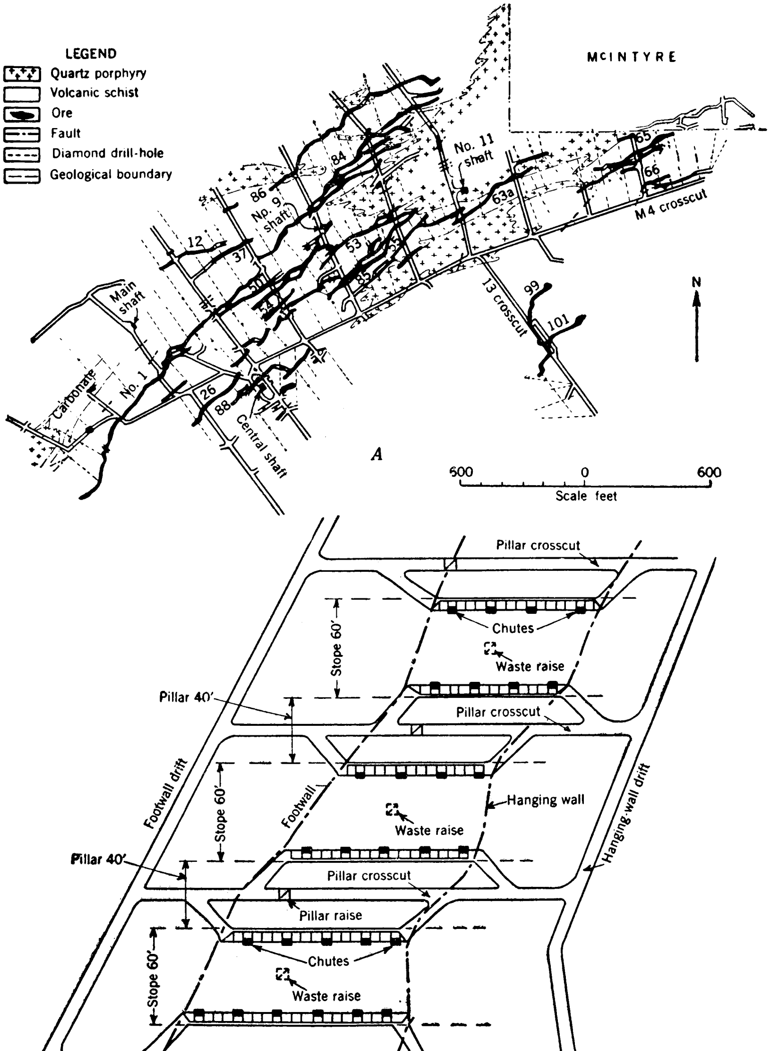
LEGEND
Quartz porphyry
Volcanic schist
Ore
Fault
Diamond drill-hole
Geological boundary
McINTYRE
No. 11 shaft
No. 9 shaft
Main shaft
Carbonate
No. 1
Central shaft
13 crosscut
M 4 crosscut
84
86
12
37
50
54
53
55
83
63a
65
66
99
101
26
88
N
A
600
0
600
Scale feet
Pillar crosscut
Chutes
Waste raise
Stope 60'
Pillar 40'
Footwall drift
Footwall
Hanging wall
Hanging-wall drift
Pillar raise
B

Figure 57.—*A*, Plan of 550 level, Hollinger mine, Ontario, 1924. (After A. G. Burrows, Ontario Department of Mines.) *B*, Part of a level, Homestake mine, South Dakota, showing scheme of development (1930).

Two main haulage drifts were driven, one in the hangingwall and one in the footwall, which were connected at 100-foot intervals by extraction crosscuts through the ore as shown.

In tabular, dipping ore bodies where the ground is heavy, the drifts may be driven in waste in the footwall and parallel to the lode to reduce the cost of drift maintenance. From the footwall drift, rock raises may then be driven to the sills of the stopes, or crosscuts may be driven to the vein and connected by short drifts for ore-extraction purposes; the latter may be abandoned as soon as the stope section above has been mined out. If, in addition, the dip is fairly flat and the stopes must be filled (cut-and-fill or square-set-and-fill stopes) one drift may be driven in the ore or in the footwall for extraction of ore from the stopes above the level and another in the hanging wall, through which waste may be hauled and dumped into the stopes below the level.

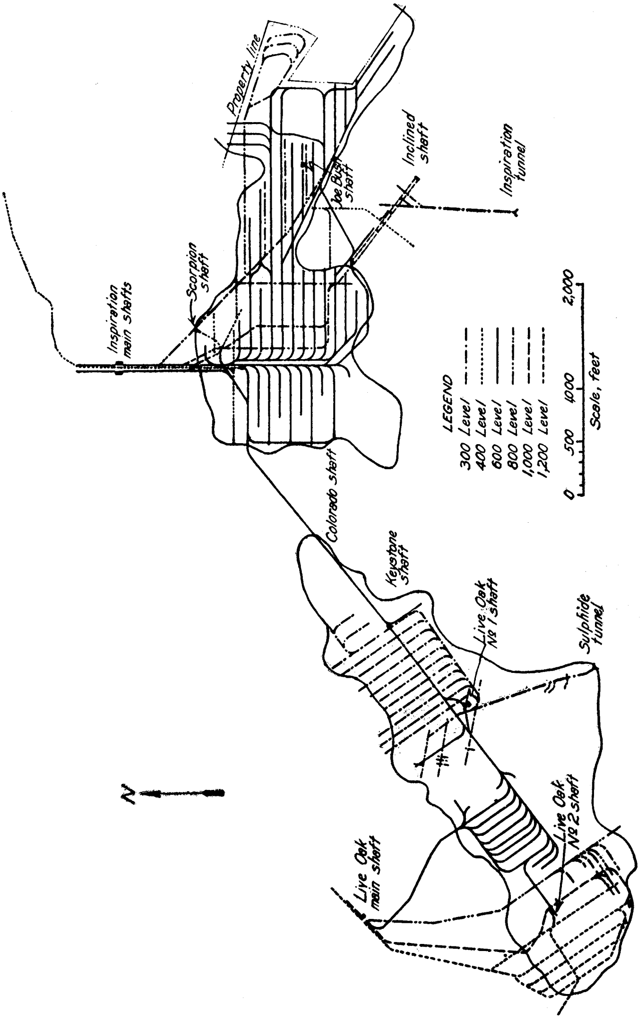
Property line
Inclined shaft
Inspiration tunnel
Joe Bush shaft
Scorpion shaft
Inspiration main shafts
LEGEND
300 Level
400 Level
600 Level
800 Level
1,000 Level
1,200 Level
0 500 1000 2,000
Scale, feet
Colorado shaft
Keystone shaft
Live Oak No 1 shaft
Sulphide tunnel
Live Oak No 2 shaft
Live Oak main shaft
N

Figure 58 shows the level-development plan (1929) of the Inspiration mine, where large bodies of low-grade copper ore of the "porphyry" type are mined by block caving. Note the equal spacing of drifts and standard curves to facilitate rapid movement of ore trains.

Other level plans or patterns differ in detail, but those mentioned are typical.

Shaft-station plans or "plats" are of two general kinds— (1) those in which a connecting crosscut or station is driven straight out from the hoisting compartments (fig. 59, *A*) and (2) those in which the drift or connecting crosscut is driven parallel to the shaft (fig. 59, *B*).

In general, the first plan is employed where the loaded mine cars are hoisted to surface on cages or where they are dumped directly into skips without intervening storage pockets, and occasionally where pockets are employed. With this arrangement, trains of cars must be uncoupled for hoisting or for dumping one at a time.

With the second plan, trains of cars may be run past the shaft and dumped, without uncoupling, into skip pockets or through short chutes directly into the skips. Where large tonnages are handled, this plan usually will facilitate more rapid and efficient ore handling.

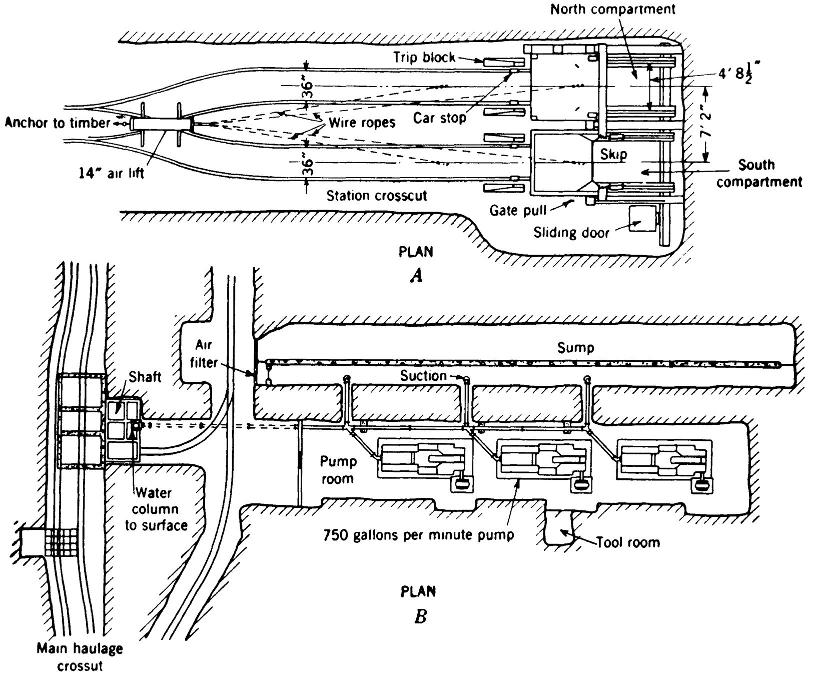
North compartment
Trip block
36"
4' 8½"
7' 2"
Anchor to timber
Wire ropes
Car stop
Skip
South compartment
14" air lift
36"
Station crosscut
Gate pull
Sliding door
PLAN
A
Air filter
Shaft
Sump
Suction
Pump room
Water column to surface
750 gallons per minute pump
Tool room
PLAN
B
Main haulage crossut

FIGURE 59.—Two general types of station plans. *A*, Crosscut straight out from shaft. *B*, Crosscut parallel to shaft.

The subject of shaft stations and skip pockets will be discussed in greater detail under "Handling and Haulage of Ore."

Pump rooms and sumps (fig. 59, *B*), car-repair shops, battery-charging stations, and locomotive barns often are situated near the shaft, and the details of the level plan will vary with their size, shape, and position.

STOPE DEVELOPMENT

The foregoing discussion relates to general mine development and covers the work required to reach and penetrate the ore bodies and to provide the main arteries through which the ore is taken from the mine. These arteries provide mine drainage and avenues for travel and transportation of men and supplies and for installation of compressed air and drilling-water and electric transmission lines.

Little or no general or stope development may be required in some types of ore bodies that have been roughly delimited by drilling before development, and where strength of ore and wall rock, size of the ore bodies, and other physical conditions are such that the ore can be stoped virtually as it is developed. Typical of this type are the mines of the Tri-State lead and zinc district.

Other conditions may require varying amounts of stope preparation before actual extraction of ore by stoping can begin.

Many mining companies combine costs of general and stope development for accounting purposes, whereas others include stope-development costs as a part of the stoping cost. In this discussion, distinction is made between general development and stope preparation. Although stope development often is done simultaneously with general development and stoping and continues after stoping has begun, it is a distinct operation and has for its purpose actual blocking out of the ore and provision of facilities for carrying on stoping operations and removal of the ore. It consists principally of raising, installing ore passes and loading chutes, and driving sublevels.

The amount and nature of the stope development required depend on the stoping method selected, which, in turn, is determined by the physical characteristics of the deposit. Thus, in preparing for shrinkage stoping, chutes are installed at rather close intervals along the drifts, when stoping may begin immediately if the stopes are silled out on the level or on top of the drift timbers. If the stopes are to be started on top of drift pillars 10 to 15 feet or more above the back of the drift, raises are extended upward from the chutes the required distance, where they are connected by small subdrifts, which are slabbed off to the walls the full width of the ore to form the stope sill (fig. 60, *A*). In either case, it usually is necessary to run at least one raise through to the level above for each stope to provide adequate ventilation and means of egress to the upper level. These through raises, however, often are run after the stope has been in operation for some time and has advanced upward a considerable distance.

In preparation for cut-and-fill stoping, chutes are spaced considerably farther apart, and usually a timbered or cribbed manway is carried alongside each chute or alternate chutes for entrance to the stope and for handling timber and supplies into the stope. With this method of stoping, it is usually necessary to run a through-raise in each stope section to the level above before stoping begins, to provide a means for introducing waste filling and for ventilation of the stope. These raises serve also as a base from which to start cutting the successive stope floors. Likewise, a through raise generally is required in square-set stoping for ventilation and filling, and chutes must be provided at the level for drawing off the broken ore.

Where stopes are silled on the level or immediately over the drift timbers, the first step in stope preparation may be the slabbing of the sides of the drift to the walls of the vein or taking down the back to make room for sill timbering and starting the stope over the timber, or both.

The time required and amount of work to be done in preparation for shrinkage, cut-and-fill, and square-set stoping are relatively small, and stoping can be begun soon after the main level has advanced a hundred feet or so along the lode. Sublevel stoping, sublevel caving, top-slicing, and block-caving require more stope preparation before ore extraction can begin.

In sublevel stoping (a form of open stoping), ore chutes, manway, and "starting raises" must be put up and one or more drifts driven on each sublevel (fig. 60, *B*).

In sublevel caving and top-slicing, raises are put up at intervals of about 20 to 50 feet or more to the top of the ore body to serve as ore passes, manways, and timber slides, and these usually are connected by a so-called timber drift on the top sublevel before slicing is begun. If the bottom of the ore is some distance above the main haulageway, so that raises from that level would require a large amount of barren rock work, it may be desirable to put up a rock raise at one or both ends of a block of ore and drive a transfer drift in ore at or just above the bottom of the ore body, from which raises are then driven in ore at regular intervals to the top of the ore. In recent years, transfer-drift systems have come to be employed in some mines, even where

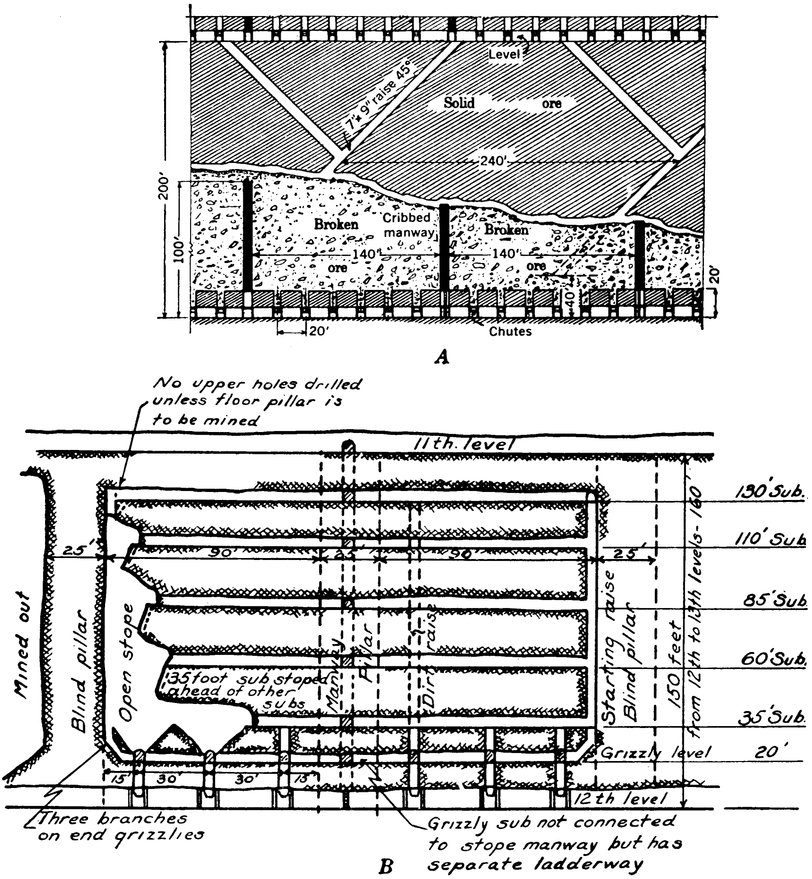
Level
7'×9" raise 45°
Solid ore
240'
200'
100'
Cribbed manway
Broken ore
140'
Broken ore
140'
20'
40'
20'
Chutes
A
No upper holes drilled unless floor pillar is to be mined
11th. level
130' Sub.
110' Sub
85' Sub
60' Sub.
35' Sub.
20'
25'
90'
25'
90'
25'
Mined out
Blind pillar
Open stope
35 foot sub stoped ahead of other subs
Manway
Pillar
Dirt raise
Starting raise
Blind pillar
150 feet
from 12th to 13th levels - 160'
Grizzly level
15'
30'
30'
15'
12th level
Three branches on end grizzlies
Grizzly sub not connected to stope manway but has separate ladderway
B

FIGURE 60.—*A*, Shrinkage stope showing chutes and raise development. *B*, Sublevel stope showing chutes, raises, and sublevel drifts.

the raises would be in ore, to cut down the cost of driving and maintaining raises.

In preparation for block caving on a large scale, considerable time is required and relatively large expenditures are involved before ore breaking by caving begins. As usually practiced, it involves installation of a series of loading chutes along each haulage drift, from each of which long, inclined, main extraction or transfer raises and branch raises are driven; the driving of grizzly drifts, cutting of grizzly chambers, and installation of grizzlies; the driving of boundary cutoff raises, subdrifts, and, in some cases, boundary shrinks; the driving of finger raises and installation of draw sets; and work on the undercutting level. (Fig. 61, *B* and *C*.) There may be some difference of opinion as to which of these operations should be included with stope development and which with stoping. Thus, there may be some question as to whether the excavation of boundary shrinks is a part of stope development or stoping, and as to whether the work on the undercutting level is stoping or preparation. The exact classification may reasonably depend upon the particular details of practice employed in specific instances. Boundary shrinks are certainly,

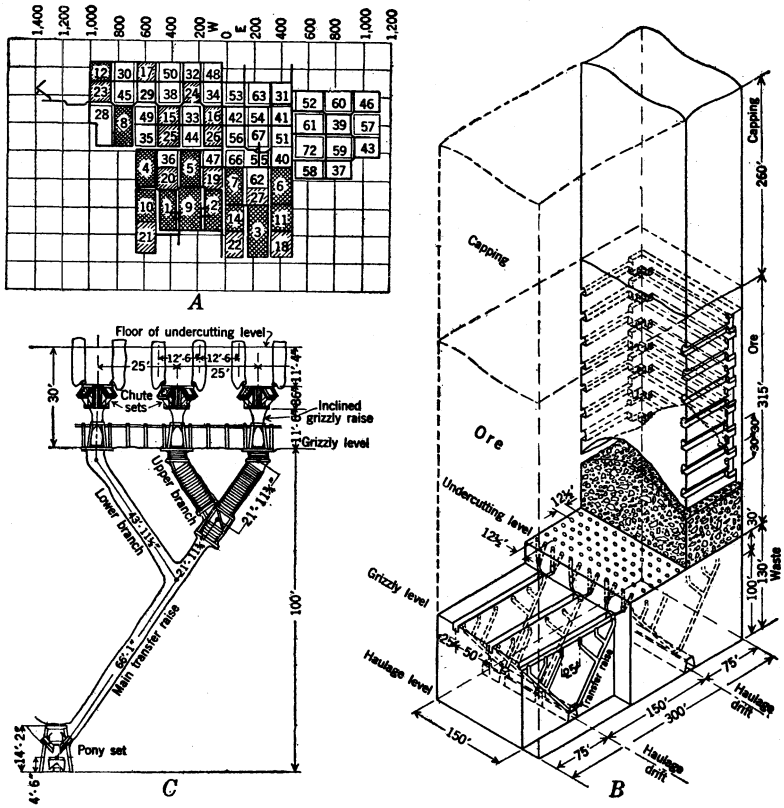
1,400
1,200
1,000
800
600
400
200
W
0
E
200
400
600
800
1,000
1,200
12
30
17
50
32
48
23
45
29
38
24
34
53
63
31
52
60
46
28
8
49
15
33
16
42
54
41
61
39
57
35
25
44
26
56
67
51
72
59
43
4
36
5
47
66
55
40
58
37
20
19
7
62
6
27
10
1
9
2
14
11
21
22
3
18
A
Floor of undercutting level
25'
12'-6"
12'-6"
25'
30'
11'-4"
11'-6"
Chute sets
Inclined grizzly raise
Grizzly level
Upper branch
Lower branch
43'-11¼"
21'-11¼"
21'-11¼"
Main transfer raise
66'-1"
100'
14'-2"
4'-6"
Pony set
C
Capping
260'
Ore
315'
30'
30'
30'
130'
100'
Waste
Capping
Ore
Undercutting level
12½'
12½'
Grizzly level
Haulage level
25'
50'
12½'
Transfer raise
150'
75'
150'
300'
75'
Haulage drift
Haulage drift
B

FIGURE 61.—Undercut block-caving, individual stope system, Miami mine, Arizona. *A*, Plan of 510-foot boundary caving level showing order of mining stopes. *B*, Isometric drawing of 150- by 300-foot stope. *C*, Detail of ore-transfer raise system.

strictly speaking, stopes, and the operation involved is stoping. The driving of drifts on, or connecting finger raises to, the undercutting level might logically be considered as development; whereas the slabbing of the drifts or the belling out of the tops of the finger raises to meet each other might be considered as either stoping or development.

The size and shape of the ore body and the stoping method employed obviously will have an important influence on the tonnage of ore developed by a given amount of development work. The tons of ore developed per foot of stope development have been estimated closely for a number of mines using various stoping methods and have been approximated for other methods. These figures are given in tables 19 to 22, inclusive. The examples selected for tabulation represent a wide range of conditions and of variations in widths and sizes of ore bodies.

TABLE 19.—*Amount of stope development required in typical open-stope mines*

Mine	Variation of open-stope method	Stope development per ton of ore developed, foot	Ore developed per linear foot of stope development, tons
Burra-Burra, Ducktown, Tenn.[1]	Sublevel stoping, 40-foot sublevel interval.	0.0083	120
Isabella, Ducktown, Tenn.[2]	Room-and-pillar; underhand stoping	.0050	200
Menominee range, mine A, Mich.[3]	Sublevel stoping	.0300	33
Menominee range, mine B, Mich.[4]	do	.0180	55
Menominee range, mine C, Mich.[5]	do	.0220	45
Menominee range, mine D, Mich.[6]	do	.0500	20
Marquette range, mine E, Mich.[7]	Overhand and breast stoping, regular pillar support, rooms 22 by 22 feet.	.0167	60
Gold mine, Ont.[8]	Sublevel stoping	.0080	125
Horne, Que.[9]	Inclined stopes benched as in sublevel stoping.	.0130	75

[1] Hard, firm ore and walls; sublevel interval, 40 feet; wide ore.
[2] Hard, firm ore and walls; pillar support; wide ore.
[3] Firm, strong, moderately hard ore; walls moderately strong; sublevel interval, 25 feet, wide ore.
[4] Ore and walls moderatly strong; sublevel interval, 25 feet.
[5] Ore moderately strong and hard; walls strong but slough after long exposure; sublevel interval, 25 feet.
[6] Ore soft and not very strong; stope faces slope outward; sublevel interval, 25 feet.
[7] Ore and walls firm; ore moderatly hard.
[8] Ore and walls hard and strong; sublevel interval, 33 feet.
[9] Ore and walls hard and strong; interval between inclines, 30 feet.

TABLE 20.—*Amount of stope development required in typical shrinkage-stope mines*

Mine	Variation of shrinkage method	Average width of ore, feet	Stope development per ton of ore developed, foot	Ore developed per linear foot of stope development, tons
Nevada-Massachusetts, Nev.	On drift pillars	4.5	0.020	50
Harmony, Idaho	On drift pillars with grizzlies over chutes.	5.0	.020	50
Hillside, Ill.	On drift pillars in wide ore; on stulls in narrow ore.	12.0	.010	100
Daisy, Ill.	On stull timbers	6.5	.008	125
Eighty-five, N. Mex.	15 percent on drift pillars; 85 percent on stulls.	5.0	.008	125
Teck-Hughes, Ont.	On drift pillars	9.0+	.0133	75
Lake Shore, Ont.	do	9.0+	.0168	60
Sylvanite, Ont.	On drift sets	4.0	.020	50
Vipond, Ont.	Partly on stulls; partly on drift pillars.	8.0+	.0045	222
McIntyre, Ont.	On drift sets	10.0	.0067	150
Engels, Calif.	On drift pillars	Wide	.005	200
Bristol, Mich.	Transverse stopes, 15 to 25 feet wide between pillars 12 to 22 feet wide. Pillars caved and drawn with shrink.	[1] 140	[2] .0028	357
Climax, Colo.[3]	Stopes 400 feet by 50 feet, separated by 40-foot pillars.	([4])	.0025	400
Alaska-Juneau, Alaska	Forced caving by large blasts in coyote drifts.	2,000+	[5] .008 [6] .005	[5] 125 [6] 200
Beatson, Alaska	Stopes 70 feet wide between pillars 30 feet wide. Large rounds drilled around mining raises.	[1] 340	.0131	76

[1] Maximum.
[2] Long tons.
[3] 1929.
[4] Hundreds of feet.
[5] Including coyote drifts.
[6] Not including coyote drifts.

TABLE 21.—*Amount of stope development required in typical sublevel-caving and top-slicing mines*

Mine	Stoping method	Stope development per ton of ore, foot [1]	Ore developed per linear foot of stope development, tons [1]
No. 16	Sublevel caving; sublevel interval 20 feet	0.0083	120
No. 19	do	.0100	100
No. 20	Sublevel caving; sublevel interval 18 feet	.0172	58
Mesabi No. 1	Parallel top-slicing, scraping into cars	.0111	90
Mesabi No. 2	do	.0101	99
Mesabi No. 3	do	.0209	48
Mesabi No. 4	Parallel top-slicing, scraping into cars (interval 14 feet)	.0278	36
Mesabi No. 5	Parallel top-slicing, scraping into cars (interval 13 feet)	.0294	34
Mesabi No. 6	Radial top-slicing, scraping to chutes (interval 12 feet)	.0242	41
Vermilion No. 7	Radial top-slicing, scraping to chutes (interval 12½ feet)	.0156	64
Cuyuna No. 8	Radial top-slicing, scraping to chutes (interval 11 feet)	.0111	90
Marquette No. 9	Parallel top-slicing, scraping to chutes (interval 11 feet)	.0143	70
Marquette No. 10	Radial top-slicing, scraping to chutes (interval 10 feet)	.0143	70
Marquette No. 11	Radial and parallel top-slicing (interval 10 feet)	.0061	164
Marquette No. 12	Parallel top-slicing, scraping to chutes (interval 11 feet)	.0088	114
Marquette No. 13	Parallel top-slicing, scraping to chutes (interval 10 feet)	.0061	164
Menominee No. 14	Parallel top-slicing, scraping to chutes (interval 12 feet)	.0260	38

[1] Long tons of 2,240 pounds.

TABLE 22.—*Preparatory work per ton of ore developed at typical block-caving mines*

Mine and kind of development	Development per ton of ore			Ore per unit of development, tons
	Linear foot	Square foot	Ton	
Ray, Ariz.:				
Supply drifts	0.00058			[1] 1,723.0
Grizzly drifts	.00260			[1] 385.0
Shrinkage stopes			[2] 0.029	[3] 34.5
Undercutting			.020	[3] 49.8
Total	.00318		.049	[1] 314.0 [4] 20.4
Miami, Ariz.:				
Grizzly level drifts	.00131			[1] 763.0
Grizzly level raises and chutes				[5] 13,917.0
Boundary drifts	.00436			[1] 229.0
Boundary corner raises	.00057			[1] 1,754.0
Finger raises	.00575			[1] 174.0
Undercutting drifts	.00125			[1] 585.0
Undercut mining		.04484		[6] 22.3
Drilling and blasting boundary drifts	.00436			[1] 229.0
Drilling and blasting corner raises				[7] 68,439.0
Total	.01760	.04484		[1] 56.8
Humboldt, Ariz.:				
Supply drifts	.00108			[1] 923.0
Grizzly drifts	.00325			[1] 307.0
Grizzlies				[8] 8,571.0
Finger raises	.00691			[1] 145.0
Pilot raises	.00067			[1] 1,500.0
Shrinkage drifts	.00133			[1] 750.0
Shrinkage stopes		.075		[6] 13.3
Undercutting		.083		[6] 12.0
Total	.01324	.158		[1] 75.5 [6] 6.3

[1] Per foot.
[2] Tons of broken swell only.
[3] Per ton.
[4] Per ton of swell.
[5] Per set.
[6] Per square foot.
[7] Per raise.
[8] Per grizzly.

STOPE-DEVELOPMENT METHODS

The foregoing section indicates in a general way the nature of work comprised in what is termed stope development. Installation of chutes for drawing off ore on the haulage levels is in reality a part of stope preparation. However, chutes are employed for loading ore into the mine cars (the first operation in transportation of ore), and details of chute construction and operation therefore will be discussed later under the caption "Handling and Transportation of Ore and Waste."

The methods of driving sublevel or sill drifts a short distance above the haulage levels are similar to those employed in driving main-level headings. They are often, but not necessarily, smaller in cross section, and, when driven to connect a line of raises, direct shoveling into raises or handling of broken ore and rock for short distances in wheelbarrows may be employed. Types of rounds similar to those already described are used.

RAISES

Raises are driven in stope development for ore passes from the stopes to the loading chutes, to provide manways for entrance to the stopes, for handling timber and supplies into and from the stopes, for ventilation, for introduction of waste filling, as bases from which to start stoping, and for installation of air and drilling-water lines and electric cables. In flat-dipping seams they may be laid with track for handling ore, timber, and supplies in cars or skips, and in some instances may actually constitute part of the general mine development.

They may be single-compartment or divided into two or more compartments, depending on their use. Likewise, the function of the raise will determine to a considerable extent its size and shape. The nature of the ground, the inclination, and the use to which it is to be put will determine whether or not it must be timbered.

Single-compartment raises sometimes are driven for ore passes or for prospecting purposes, but two-compartment raises in which one compartment serves as an ore pass and the other as a manway are more generally employed. If ore hangs up in a long single-compartment raise, great difficulty and danger often are experienced in clearing it; whereas, if there are two compartments, the trouble can be attacked from the manway at the point where the hang-up occurs by making an opening in the side. In some mines, described later, three-compartment raises are standard for certain purposes.

Raise timbering may consist of stulls, cribbing, sets constructed similarly to shaft sets, or square sets with lagging and lining between the compartments as required.

In starting a raise, it is customary to carry it up several rounds before installing chutes or permanent timber if the ground will stand, to avoid damage to the timber from blasting and to keep the top of the timbered section some distance below the face as the raise is advanced upward. On the other hand, it is advantageous in a two-compartment raise to install the loading chute as soon as possible to eliminate shoveling of the broken rock.

Whether or not the raise is to be timbered, a temporary staging is necessary in steep raises, from which to reach and drill the back, and means of access thereto must be provided. If the raise inclination is less than about 55° or 60°, the miners can walk up on the footwall, with the aid of a rope, if necessary. For steeper inclinations, chain or rope ladders often are employed, but it is usually best to install the regular timbering with fixed ladders to within a short distance of the back as the raise advances. In some localities the mining laws require that raises steeper than 55° and more than a given maximum length shall have at least two compartments and shall have a ladderway equipped with suitable ladders.[6] Such regulations usually specify the maximum allowable distance from the top of the last timber to the face. To avoid damage to the timbers from blasting, the ore pass or chute compartment is usually kept nearly full of broken ore, and a bulkhead (fig. 62, *A*) is constructed over the manway or it is covered with pieces of short timber before blasting (fig. 62, *B*).

Figure 63 shows two- and three-compartment stull raises, figure 64 a two-compartment cribbed raise, and figure 65 a two-compartment square-set raise.

The type of drill rounds employed in drilling raises are in principle the same as those used in drifts, crosscuts, and shafts. Thus, center V-cut, side-V or draw cut, and burnt-cut rounds are all employed. The pyramid-type cut is sometimes used in large raises. Figure 66 illustrates typical raise rounds. At *A* there is shown a stoper drill

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[6] Handbook of Regulations Governing the Operation of Mines: Part VIII and Sections 178, 179, and 180 of Part IX of the Mining Act of Ontario, 1930, p. 39.

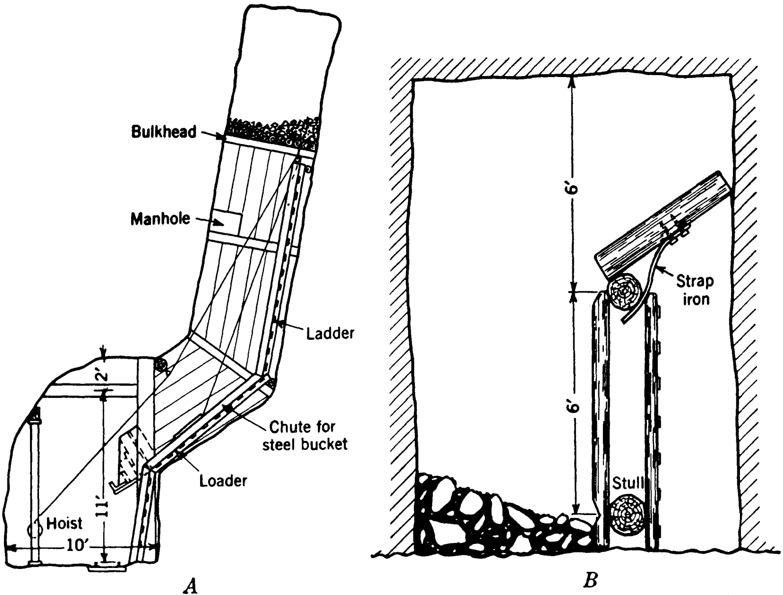

Bulkhead
Manhole
Ladder
Chute for steel bucket
Loader
Hoist
2'
11'
10'
A
6'
6'
Strap iron
Stull
B

FIGURE 62.—*A*, Manway protected by bulkhead. *B*, Timbered raise, manway protected by timber prior to blasting.

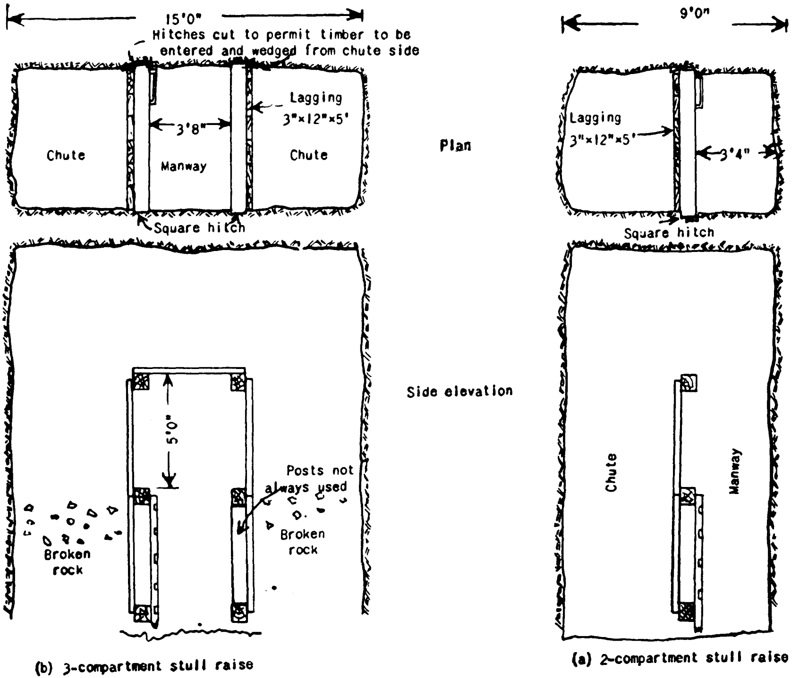
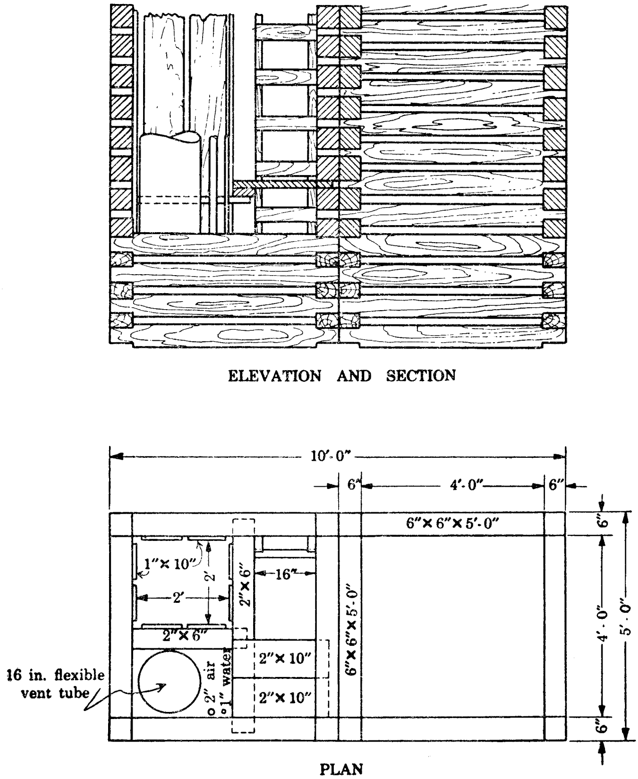

15'0"
Hitches cut to permit timber to be entered and wedged from chute side
Lagging 3"×12"×5'
3'8"
Chute
Manway
Chute
Square hitch
Plan
9'0"
Lagging 3"×12"×5'
3'4"
Square hitch
5'0"
Posts not always used
Broken rock
Broken rock
Side elevation
Chute
Manway
(b) 3-compartment stull raise
(a) 2-compartment stull raise

FIGURE 63.—Stulled raises.

set up on staging in an inclined raise. It is apparent that in small square or short rectangular raises, pyramid or center V cuts are difficult to drill because the foot of the stoper leg or the back end of a jackhammer or drifter-type machine cannot be swung back far enough to give the required inclination to the holes. The burnt cut (fig. 66, *B*) has an advantage in this respect in small square raises because the holes are drilled straight in, normal to the face. In a rectangular raise 7 feet or more in length, fairly deep V-cut rounds may be drilled readily but cutting the V at one end (draw cut; fig. 66, *C*). Figure 66, *D*, shows a center V-cut round in a 5 by 7 foot raise.


ELEVATION AND SECTION

PLAN

FIGURE 64.—Two-compartment cribbed raise.

Except in very hard ground, stoper machines usually are employed in vertical or steeply inclined raises. In exceptionally hard ground heavy drifter-type machines mounted on a cross bar often are employed.

Raises, especially long ones, often are difficult to ventilate, and unless some form of forced ventilation is employed, considerable time may be lost after blasting waiting for smoke to clear before the miners can ascend to the face in safety. In the past, gassy raises have too frequently caused fatal accidents. In two-compartment raises a fan pipe may be carried in the manway if it is large enough and a fan be employed to blow air to or exhaust gas from the face.

Sometimes advantage may be taken of exploratory drill holes and raises may be driven along them, a compressed-air ejector being employed at the top of the hole on surface or on the level above to draw out blasting fumes and provide ventilation. In some instances, it may even pay to drill a long hole along the proposed course of the raise for the sole purpose of providing ventilation. It is surprising how much air can be drawn through even a 1¼- or 1½-inch hole in a short time by this method, which has been employed successfully at a number of mines. Figure 67 shows an ejector employed at the Bussières mine in Quebec[7] and the method of installing it on an upper level. In this instance several raises were run simultaneously on drill holes put down from the same drift and the ejectors discharged into a common ventilating pipe connected to an exhaust fan.

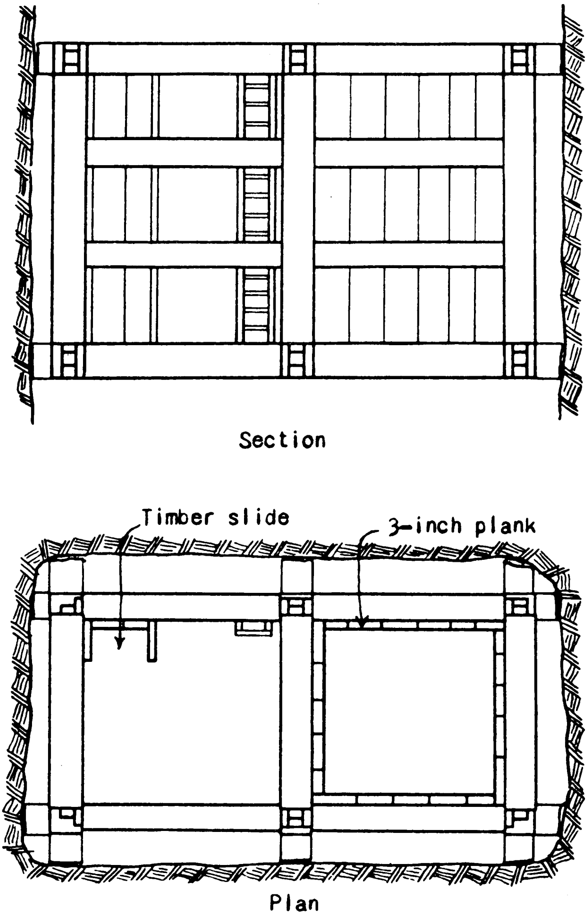


FIGURE 65.—Raise timbered with square sets.

COST OF DRIVING RAISES

Table 23 presents typical raising costs and includes data on the sizes of raises, types of rounds employed, and methods of timbering.

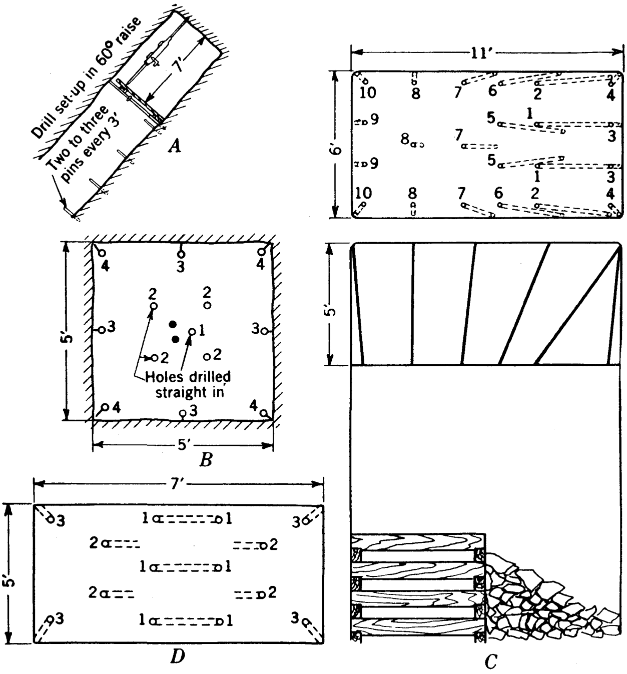

Drill set-up in 60° raise
Two to three pins every 3'
7'
A
11'
6'
10 8 7 6 2 4
9
5 1
8 7 3
9 5
1 3
10 8 7 6 2 4
5'
4 3 4
2 2
3 1 3
2 2
Holes drilled straight in
4 3 4
5'
B
5'
7'
5'
3 1 1 3
2 2
1 1
2 2
3 1 1 3
D
C

FIGURE 66.—Types of raise rounds (figures indicate order of firing). *A*, Stoper drill set up on staging. *B*, Burnt cut. *C*, Draw cut. *D*, Center V cut.

The cost of raising increases with length or height of raise after the first 50 feet or so, and when the work is done on contract a sliding scale of rates usually is established, which provides one price for the first 25 or 50 feet, the rates increasing by increments with increased distances driven.

Table 24 gives costs of raising in terms of labor and unit-supply consumption.

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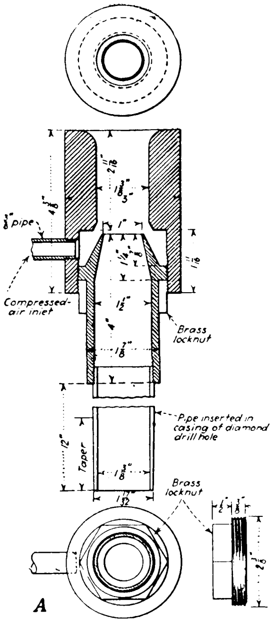
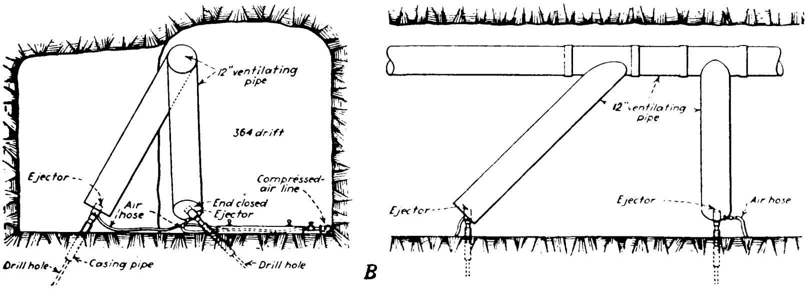
[7] Jackson, Chas. F., Ventilating Raises and Stopes Through Boreholes: Eng. and Min. Jour., vol. 136, No. 6, June 1935, pp. 284–285.


1/8" pipe
Compressed-air inlet
Brass locknut
Pipe inserted in casing of diamond drill hole
Taper
12"
Brass locknut
A

This air ejector is connected to the collar of the diamond-drill hole at the 350 level


12" ventilating pipe
364 drift
Ejector
Air hose
End closed
Ejector
Compressed-air line
Drill hole
Casing pipe
Drill hole
B
12" ventilating pipe
Ejector
Ejector
Air hose

Cross section of drift

Longitudinal projection

Gases pulled out of the 500-level workings are ejected into this ventilating pipe in the 364 drift, which pipe discharges into the shaft. This drawing shows the way in which connection with the ventilating pipe is made. With normal air pressure of about 85 pounds the suction created rapidly removes the gases from raise or stope

FIGURE 67.—Air ejector and method of installation at Bussières mine, Quebec.

TABLE 23.—*Typical raising costs*

Mine	Nature or kind of rock	Size of raise, feet	Inclination	Drill round	Timbering	Costs per foot of raise					
						Direct labor and supervision	Explosives	Timber	Other supplies and power	Haulage, hoisting, and miscellaneous	Total
Hanover-Bessemer, N. Mex., 1930.	Iron ore (magnetite) and limestone.	7 by 10	55° to 70°		10 inch by 10 inch stulls.	$2.54	$1.52	$0.13	$1.29		[1] $5.48
Teziutlan, Mexico, 1930-31.	Massive sulfide and metamorphosed sediments, hard and firm.	5 by 8			8-inch by 8-inch stulls.	4.40	1.25	.73	3.54	$2.71	12.63
Engels, Calif., 1930	Hard, tough diorite	6 by 8	60° plus	14 to 18 holes, pyramid cut.	Stulls						18.34
Eighty-Five, No. 1780 raise, N. Mex., 1930.	Quartz vein	5 by 9			do	8.60	2.97	.82	[2] 5.16		17.55
Elkoro, Nev., 1931	do	Width of ore by 10 to 20.	60° to 80°		do	5.46	1.07	1.05	[3] 1.69	[4] 1.03	10.30
Mt. Hope, N. J., 1931	Iron ore and gneiss	5 by 6	45° to 65°	22-hole, burnt cut	None	3.73	1.42	.06	2.93	.26	8.40
Atlanta Idaho, 1934.	Quartz-caldite vein in granite.	3.5 by 7	70° to 85°	12 holes	Framed sets	4.50	(5)	(5)	(5)		[6] 7.57
Hollinger, Ont., 1934 [7]	Quartz veins in schist; hard.	5 by 6 to 6 by 7	45° to 50°	Burnt cut, 15 to 18 holes.		6.11	1.58		1.50		[8] 9.19
Ashley, Ont	Quartz vein in basaltic schist.	5 by 10	50° to 55°			6.79	3.68		7.04	[9] 5.85	[10] 23.36
Eustis, Quebec [11]	Sericite schist and pyrite ore.	5 by 5	60° plus		None	3.08	3.30	.21	2.02	.40	9.01
Pioneer, B. C. [12]	Quartz vein, greenstone, diorite.	4 by 14	78°	18-hole center V-cut	Stulls	5.94	3.06	.65	2.60	.50	12.75

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[1] Haulage and hoisting apparently not included; direct costs only.

[2] Includes $5.05 expense operating rock drills.

[3] Includes $1.24 expense of operating rock drills.

[4] Includes outside expense $0.35, assaying $0.112 and general expense $0.565.

[5] All supplies and power, $3.07.

[6] Direct cost only.

[7] Young, Arthur W., Mining [Hollinger Consolidated Gold Mines, Ltd.]: Canadian Min. Jour., September 1935, p. 379.

[8] Direct cost only, but includes compressed air, rock drills, and drill steel.

[9] Includes rock drills, drill steel, pipes and fittings, boiler fuel, oil, gasoline, etc.

[10] Includes blacksmithing, decking and surface tramming, teaming and trucking, and camp expense.

[11] Snow, Fred W., and Brownbill, H. F., Mining Methods and Costs at the Eustis Mine: Canadian Min. and Met. Bull. 286, February 1936, Transactions Sec., p. 81.

[12] Fawley, Allen P., Mining Gold Ore at Pioneer: Canadian Min. and Met. Bull. 305, September 1937, Transactions Sec., pp. 475–486.

TABLE 24.—*Labor, supplies, and power per foot of raise*

Mine	Nature or kind of rock	Size of raise, feet	Inclination	Drill round	Labor, man-hours per foot			Explosives, pounds per foot	Timber per foot	Power, kw.-hr. per foot
					Direct	Other	Total			
Burra-Burra, Tenn	Massive sulfide, schist, graywacke.	5 by 5	55° to vertical	Burnt cut	4.4			8.8	None	27
Spring Hill, Mont	Hard and tough	6 by 8	60°	Side V or draw	6.83			10.2	do	43
Do	do	6 by 9	60°	Center V	8.35			12.0	19.2 bd. ft	44
Teziutlan, Mexico	Massive sulfide, metamorphosed sediments, hard and firm.	5 by 8			19.45	15.34	34.79	5.48	26.2 bd. ft	189
Kirkland Lake Gold, Ontario.	Hard; syenite, porphyry, and lamprophyre.	5 by 6	45° to 50°	Pyramid cut	12.3	[1] 6.6	18.9	20.0	None	[1] 260
Elkoro, Nev	Quartz vein	Width of ore by 10 to 20 feet.	60° to 80°		5.51	[2] 2.72	8.23	6.27	6.0 lin. ft., 10.0 bd. ft.	
Morning, Idaho	Quartzite, hard, swelling ground	5 by 10	45°				[3] 11.80	14.0	68.3 bd. ft	
Champion, Mich	Brecciated basaltic lavas; hard	4 by 8	70°	Burnt cut, 26 holes	6.6	1.1	7.7	14.0	8.0 bd. ft	
Black Rock, Mont	Vein and granite; hard	8 by 14	Vertical	Side draw or V cut, 15 to 18 holes.			[3] 15.1	11.8	101.0 bd. ft	
Page, Idaho	Quartzite, soft and loose	6 by 11	40° to 60°	Side draw, 16 holes			[3] 4.8	13.6	195.0 bd. ft shaft sets.	
Montreal, Wis	Slates and iron formation, medium hard.	4 by 8 cribbed	65°	10 holes	13.03			13.67	60 bd. ft	
Do	do	4 by 5	65°	4 holes	6.38			8.61	None	
Do	Hematite ore, soft	4 by 8	65°	10 holes	8.16			6.1	60 bd. ft	
Do	do	4 by 5	65°	4 holes	3.23			6.0	None	
Eureka-Asteroid, Mich	Hematite, medium soft	5 by 9 cribbed	55° to 90°		8.45	1.70	10.15	7.0	27 bd. ft	
Do	Medium hard slates and iron formation.	do	do		14.83	1.70	16.53	12.0	27 bd. ft	
Old Dominion, Ariz	Mostly vein matter	5 by 9 cribbed	35° to 90°		6.56			6.01	120 bd. ft	
Braden, Chile	Andesite porphyry, medium hard.	6.5 by 6.5	60°	16 holes	2.10			4.32	None	

[1] Includes hoisting.

[2] Haulage, hoisting, and supervision.

[3] Includes haulage, hoisting, and supervision.

SUMMARY OF MINE DEVELOPMENT

The principal steps in preparing a mine for ore extraction and the unit costs of the various operations involved have been discussed briefly. It has been pointed out that seldom, if ever, is a mine fully explored and developed before stoping operations are begun, but that exploration and development continue during stoping. Under most conditions, to explore and develop a mine completely before beginning production would tie up too much capital for an unnecessarily long time, and the maintenance cost of inactive workings usually would be excessive.

To extend the productive life of a mine, new ore must be found and developed to replace that mined. The development done before ore extraction is begun must then be supplemented by continuing such work throughout the active life of the mine. To maintain a uniform rate of production over a period of years and avoid wide variations in production costs due to abnormal development charges, it is good practice, where possible, to set aside a lump sum each year for exploration and development or to allocate so much per ton mined for that purpose. If the tonnage to be mined and the cost of finding and developing an equal tonnage could be known in advance, the amount necessary to be allocated to maintain the ore-reserve position could be determined accurately. Since this is rarely possible, the operator must rely largely on estimates based upon past experience.

In numerous instances in well-established, long-life mines, ore reserves and rate of production have been maintained with a fairly uniform development-cost-per-ton charge over a number of years. This is simplified, of course, where the ore bodies are large and persistent. It becomes increasingly difficult where ore bodies are irregular, disconnected, and lack persistence; in such instances, wide fluctuations in development costs can scarcely be avoided.

COST OF MINE DEVELOPMENT

Table 25 presents typical costs of current mine development per ton of ore produced at a number of producing mines.

Attention is called especially to data on the Lamaque mine in Quebec, which show high costs incurred during the early life of a gold mine, where the desire to accumulate enough ore reserves to warrant considerable expansion of mill capacity resulted in an abnormally large amount of development work relative to current production. During the first year production was principally of ore from development headings and shaft sinking progressed at the same time.

Major development programs involving the sinking of large shafts or extensive lateral development naturally will be reflected strongly in the development costs per ton of ore produced unless, as is often done, development is carried as a deferred account to be charged off at so much per ton or so much per year over the life of the mine or over a considerable period of time. On the other hand, many operators prefer to charge off major development expenditures as they accrue, and although this may result in widely fluctuating production costs as they appear on the books, it at least has the advantage of insuring that development charges will be wiped out before the ore reserves are exhausted.

TABLE 25.—*Typical costs of current exploration and development per ton of ore produced*

A. GOLD MINES

Mine and year	Type of ore deposit	How developed	Stoping method	Tons ore mined during year [1]	Cost of exploration and development per ton mined
Homestake, S. Dak.:					
1929	Large, wide	Vertical shafts	Shrinkage, caving, square setting	1, 437, 935	$0. 130
1930	do	do	do	1, 364, 456	. 181
Alaska-Juneau, Alaska:					
1928	Stringer lodes in wide shear zone	Adit	Modified shrinkage and forced caving	3, 670, 910	. 061
1929	do	Adit and winze	do	3, 836, 440	. 297
1930	do	do	do	3, 924, 460	. 256
Elkoro, Nev., 1929	Veins	do	Shrinkage, cut-and-fill	57, 539	1. 298
United Eastern, Ariz., Jan. 1917 to May 1925.	Wide veins in andesite	Vertical shafts	Cut-and-fill	732, 528	. 518
Teck-Hughes, Ontario.					
1931	Vein in porphyry and lamprophyre	Vertical shafts, levels 125 feet	Shrinkage	396, 200	1. 180
1933	do	do	do	474, 500	1. 040
1934	do	do	do	474, 700	. 930
1935	do	do	do	383, 958	. 865
1936	do	do	do	364, 299	. 497
Lamaque, Quebec.					
1935	Flat lenses in granodiorite	Shafts	Open stope	83, 847	[2] 4. 710
1936	do	do	do	201, 356	[3] 2. 360
Lake Shore, Ontario: 1931	Two main veins 300 ft. apart	Vertical shafts, levels 150 ft	Shrinkage and cut-and-fill	698, 624	[4] 1. 403
Hollinger, Ontario,[5] 1934	Numerous veins and lenses in wide shear zone.	Vertical shafts, levels 125 and 150 feet	Cut-and-fill, some shrinkage	1, 900, 490	. 798
McIntyre, Ontario, 1931	do	Vertical shafts, levels 100 or 125 feet	do	557, 104	. 655
Wright-Hargreaves, Ontario, 1930.	Several veins in porphyry	Vertical shafts, levels 125 feet	Shrinkage and stulled stopes	220, 430	1. 072
Vipond, Ontario, 1930	Veins in sheared basaltic schists	Vertical shafts, levels 100–133 feet	Shrinkage	104, 381	. 728
Cortez (silver), Nev., 1929	Dikes, bedding planes, and fissures	Adits and inclined shaft	do	43, 806	1. 427

B. LEAD AND ZINC AND LEAD-SILVER MINES

Mine and year	Type of ore deposit	How developed	Stoping method	Tons ore mined during year [1]	Cost of exploration and development per ton mined
No. 8, Southeast Missouri, 1928	Irregular, flat, bedded deposits	Vertical shafts	Open stopes	168, 089	$0. 045
Mascot No. 2, Tenn., 1929	Large, irregular bedded deposits, dip 18° to 22°.	Vertical shaft and inclined winzes, level interval 50 to 75 feet.	Open stopes. Underhand mill holes, heading-and-bench.	528, 626	. 053
Page, Idaho, 1928	Two fissure veins in shear zone; dip 40° to 60°.	Inclined shaft, level interval 300 ft	Square sets	90, 460	1. 170
Morning, Idaho, 1928	Large, persistent fissure	Adit and shaft, level interval 200 ft	Stringer-set and fill	276, 890	. 442

Hecla and Star, Idaho, 1928	Several ore bodies in shear zone; dip 70° to 80°.	Adit and deep shaft, level interval 300 ft.	do	312, 942	. 638
Silver King Coalition, Utah, 1929	Large, irregular, bedded deposits in limestone; some veins in quartzite.	Adits, main shaft, and five auxiliary underground shafts, 200-ft. levels.	Square sets	180, 208	2. 086
Park-Utah, Utah, 1928	Fissure veins. Ore shoots 3 to 80 ft. wide, dip 40° to 55°.	Adit and shafts. Level interval 200 ft	Square sets; some cut-and-fill	192, 250	. 828
Pecos, N. Mex., 1927–29	Irregular lenses in shear zone	Two adits and 2 shafts, level interval 100 ft.	Cut-and-fill, and square set	578, 658	. 859
El Potosi, Mexico:					
1926	Irregular replacements in limestone; both pipe and tabular forms.	Shafts, levels 165 feet	Large open stopes	121, 848	. 736
1928	do	do	do	316, 840	. 748
1930	do	do	do	424, 363	. 503
1932	do	do	do	441, 094	. 269

C. COPPER MINES

Burra-Burra, Tenn., 1928	Wide vein; dip 75° to 35°	Shafts, level interval 196 ft	Sublevel stoping, open stopes	473, 292	$0. 229
Mary, Tenn., 1928	Irregular, replacement of folded beds, dip 65°.	Shaft, level interval 100 ft	Open underhand stopes	108, 519	. 355
Engels, Calif., 1928	Shear zones in diorite; dip 80°; main body 100 by 830 ft. maximum.	Adits and shafts. Level intervals 100 to 200 feet.	Shrinkage	303, 301	. 479
Matahambre, Cuba, 1928	Irregular, lenticular pipes, dip 40° to 45°.	Shaft, level interval usually 100 ft., two 130 ft.	Cut-and-fill	364, 746	. 711
Colorada (Cananea), Mexico, 6 mo., 1929.	Irregular, large ore bodies in a pipe	Shafts, level interval 100 feet	Cut-and-fill; square-set; shrinkage	262, 278	. 561
Pilares, Mexico, 1929	Vertical, tabular ore bodies and irregular masses.	do	Cut-and-fill, some square-set and shrinkage.	835, 036 mined, 696, 994 hoisted	. 399
Champion, Mich., 5 mo. ended Dec. 31, 1930.	Lode 6 to 50 ft. wide, dip 70°	Shafts, level interval 100 ft. on dip	Cut-and-fill	193, 597	. 490
Magma, Ariz., 1928	Vein, dip 45° to 80°. Up to 20 or 30 ft. wide.	Shafts, levels 250 to 300 ft	Cut-and-fill and square sets	263, 094	. 736
United Verde Extension, Ariz., 1928.	Pipelike mass tapering downward	Shafts and main tunnel, levels 100 ft	Square sets	275, 212	. 611
Morenci, Ariz., 1929. Estimated for 120,000-ton block	Large "porphyry" type deposit	Shafts and adit levels at 192 and 461 ft. below adit.	Block caving	120, 000	. 303
Ray, Ariz., 1928	Large bodies of mineralized schist	Shafts. Level interval 150 ft	do	3, 243, 159	. 1036
Estimated for 560,000 ton-block.	do	do	do	560, 000	. 146
Inspiration, Ariz., 1928	Large "porphyry" type deposit	Shafts, levels at 400, 600, and 850 feet	do	4, 879, 646	. 211
Teniente (Braden), Chile, 1928	Large crescent-shaped "porphyry" deposit.	Adits	do	5, 174, 017	. 046

See footnotes at end of table.

TABLE 25.—*Typical costs of current exploration and development per ton of ore produced*—Continued

C. COPPER MINES—Continued

Mine and year	Type of ore deposit	How developed	Stoping method	Tons ore mined during year	Cost of exploration and development per ton mined
Britannia, B. C.:					
A. East Bluff, 12-month period.	Wide bodies in shear zone dipping 65°	Adits and shafts	Undercutting and breaking down by large blasts in subdrifts.	398,664	$0.080
B. Fairview			Undercutting and breaking down by large blasts in subdrifts and glory hole.	1,268,234	.008
C. Fairview			Square set	133,195	.131
Miami, Ariz., Oct. 1, 1925, to Sept. 30, 1929.	Large "porphyry" type deposits	Shafts	Block caving	16,556,296	.100

D. IRON MINES

Mine and year	Type of ore deposit	How developed	Stoping method	Tons ore mined during year	Cost of exploration and development per ton mined
Mineville, N. Y., 1927	Beds 3 to 40 ft. thick, dipping 20° to 30°.	Vertical shafts and numerous auxiliary inclines.	Open stopes and pillars	692,726	$0.103
Mine No. 1, Marquette range, Mich., 1928.	Large, irregular masses of hard specular hematite.	Vertical shafts, level interval 40 to 50 ft., not all connected to shaft.	do	420,000	.218
Mine No. 2, Marquette range, Mich., 1928.	Many lenticular masses, sheets, and chimneys of soft ore.	Vertical shafts. Levels at 200 to 250-ft. intervals.	Sublevel (open) stoping and top slicing.	360,000	.251
Mine No. 1, Menominee range, Mich., 1928.	Large, irregular, steeply dipping masses.	Vertical shaft. Levels at 200-, 300-, 400-, 950-, and 1,200-ft. depths.	Sublevel (open) stoping	200,000	.211
Mine No. 4, Marquette range, Mich., 1929.	Lens of semihard ore, 10 to 60 ft. thick, dip 45°.	Vertical shafts. Level interval 100 feet.	Shrinkage with pillars	117,224	.237
Mine No. 5, Marquette range, Mich., 1929.	Large, irregular trough ore body up to 100 ft. thick.	do	Top slicing	555,919	[6].052
Eureka-Asteroid, Mich., 1929	Irregular bodies in pitching troughs, steep dip to flat.	Vertical shafts. Level interval 200 feet.	Sublevel caving	401,680	.317
Montreal, Wis., 1928	Irregular bodies in pitching troughs; steep dip to flat; large ore bodies.	Vertical and inclined shafts. Level interval, 200 feet on dip (65°).	do	450,000	.194
Do	do	do	Sublevel and overhand open stoping	630,000	.367
Mt. Hope, N. J.	3 large beds dipping 60° to 90°	Inclined shaft. Level interval 160 to 260 feet.	Shrinkage	176,474	[7].402

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[1] Ore mined from iron mines is given in long tons (2,240 pounds).

[2] Just entering production; development cost abnormally high.

[3] Developing ore at much faster rate than extraction.

[4] Includes shaft sinking, 1,080 feet; drifting, 11,203 feet; crosscutting, 4,330 feet; raising, 13,878 feet; shaft raising, 304 feet; ore passes, 3,783 feet; winze, 795 feet; and station cutting, 121,891 cubic feet.

[5] Young, Arthur W., Mining [Hollinger Consolidated Gold Mines, Ltd.]: Canadian Min. Jour., September 1935, p. 398.

[6] Sublevel development apparently charged to stoping and not included.

[7] Incline development, $0.104; stope development, $0.298.

TRANSPORTATION OF ORE AND WASTE

For the purpose of this discussion, "transportation" signifies the operations involved in transporting ore and waste from the face or stope where it is broken to the surface but not the handling of the ore within the stopes, which will be discussed under Stoping. It therefore includes loading cars from chutes on the haulage and intermediate levels, hand-tramming, animal and locomotive haulage, scraper and conveyor-belt transportation, operation of main transfer raises between levels, dumping into shaft pockets, the operation of skip-loading devices, the caging of cars, and hoisting the ore in buckets, cages, and skips.

Transportation facilities of some sort must be provided when underground work is begun. These may or may not be retained as mine development expands and ore production begins, but they are likely to prove inadequate after the mine comes into part or full production. Thus, hand tramming and hoisting in buckets or in small cars on cages may serve when tramming distances are short and the amount of ore and rock to be handled is small, while large cars, locomotive haulage, and hoisting in skips may be required for economical transportation once the mine hits its production stride.

The distances material must be hauled and the amount to be handled are of first importance in deciding upon the method of transportation and the kind of equipment to be used. Facilities for handling ore and waste between the stopes and the haulage levels will vary with the stoping method employed as well as with the amount of ore and waste to be handled.

In an earlier Bureau of Mines publication[8] it was pointed out that the cost of underground transportation constitutes a considerable percentage of the total mining cost in most mines, and in some mines may even exceed the cost of stoping. At 19 open-stope mines, transportation averaged 26.3 percent of the total underground cost; at 19 mines employing shrinkage stoping, 22.65 percent; at 3 mines employing block caving, 21.03 percent; at 8 mines employing square-set stoping, 12.77 percent; and at 4 mines employing cut-and-fill stoping, 12.9 percent. The average haulage and hoisting costs for these groups ranged from \$0.1135 to \$0.4025 per ton of ore mined, and the range for 47 individual mines was from \$0.086 to \$1.848. Elsing[9] has tabulated transportation costs at 66 mines, which show averages by groups ranging from \$0.120 per ton mined to \$0.556. It is therefore apparent that underground transportation merits the careful attention of the operating staff.

TRAMMING AND HAULING

HAND TRAMMING

Hand tramming is still widely used in small mines, in prospecting operations, and as an adjunct to mechanical haulage systems. For tramming small tonnages of rock and ore short distances on the main levels, for short transfers on sublevels, and in some types of stopes it is the obvious method to employ, although, if power is available, scrapers may be more economical for some of these purposes and small storage-battery locomotives for others, provided the tonnage to be handled warrants the capital outlay involved.

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[8] Jackson, Chas. F., Some Notes on Underground Transportation in Metal Mines: Bureau of Mines Inf. Circ. 6326, 1930, 40 pp.

[9] Elsing, Morris J., Cost of Mine Transportation; Eng. and Min. Jour., vol. 134, No. 3, March 1933, p. 101.

Hand tramming requires a minimum capital investment for equipment and for this reason, among others, is particularly adapted to prospects and small-scale operations such as those conducted by many "leasers" (lessees).

Small cars of 12-, 16-, or 20-cubic-foot capacity are usually employed, the 16- and 20-cubic-foot sizes being the most popular, although cars holding 2 to 2½ tons sometimes are used. In the Tri-State lead and zinc district, "cans" of 1,200 to 1,400 pounds capacity were used almost exclusively for many years, but more recently these have been superseded by cars in some of the mines. The cans are buckets that are run to the face on a flat truck and, after they have been loaded, are returned to the shaft and hoisted directly to the surface.

Cars are usually of either the end- or side-dumping type, with the end or one side, as the case may be, forming a swinging door suspended from a rod at the top of the car and latched at the bottom. In the end-dump type the car body or box often is mounted on a turntable on top of the truck frame, so that the car may be turned for dumping into a chute at either side of the track.

Car bodies and truck frames may be of either wood or steel construction, though few wood cars have been employed in recent years. One advantage of the wood-box car is that if a board becomes broken it can readily be removed and replaced, whereas a damaged steel box requires considerable work in the shop.

For tramming ore and filling material in cut-and-fill stopes, a car with a scoop-shaped box open at the front end may be used to advantage.

Tracks for hand tramming usually are laid with light rail (weighing 8, 12, or 16 pounds per yard) on wood ties. The heavier rails are preferable if they are to be used for an appreciable time, since once they are well-laid they are less likely to become bent and out of alinement and thus will require less attention.

Track grades for most efficient hand tramming should be considerably steeper than for locomotive haulage. The ideal grade would probably be that on which the effort required to push a loaded car downgrade would equal that required to push the empty car up, although some prefer a somewhat steeper grade, so that the loaded car, when once started, will almost if not quite run by itself. The exact grade requiring a minimum of effort will vary with the size and weight of the car and the load carried, the type of bearings, lubrication, and the general condition of the equipment and track. Ordinarily, grades of 1 to 1½ percent in favor of the loaded cars will be found satisfactory for hand tramming.

ANIMAL HAULAGE

Mules and horses continue to be employed for underground haulage, though much less extensively than formerly. The growing availability of electric power, the introduction of Diesel generating plants in sizes and of capacities to suit almost any condition, and the development of small storage-battery and trolley locomotives have favored the substitution of mechanical for animal haulage.

Animals are useful for gathering loaded cars from the loading faces to a siding on the main haulageway and returning the empties, particularly if large cars are employed or there are unavoidable grades to be negotiated that are too difficult for hand tramming but not steep enough for rope haulage. Cars may be handled one at a time or be made up into trains of two, three, or more, depending on conditions.

As a rule, mules are preferred to horses for underground work, and for heavy work animals weighing around 1,200 pounds are more satisfactory than smaller ones, provided there is enough head room. Smaller animals, of course, must be employed where head room is low or larger ones cannot be taken down the shaft. A cage compartment usually will accommodate a 900-pound mule.

According to van Barneveld,[10] the economical limit of mule haulage is between 1,000 and 1,500 feet, and a mule should work at a speed not to exceed 5 miles per hour when taking a loaded train down a grade that needs only tractive effort to start, or 2 to 3 miles per hour when pulling a heavy load.

Van Barneveld cites data presented by Boericke [11] regarding track grades and ton-miles per mule, as follows:

Length of grade, feet	*Grade, percent*	*Number of cars [1] pulled by one mule*
55	6. 6	2
35	15. 0	1
88	8. 7	2
34	12. 8	1
31	14. 3	1
130	7. 0	1 or 2

[1] 16.5-cubic-foot cars, roller bearings, weighing 800 pounds, with net load of 1,500 pounds.

The following table from Boericke's article indicates performance of mules under different conditions:

TON-MILES OBTAINED UNDER VARYING CONDITIONS

A. Grades against the load. Maximum, 300 feet, 3 percent. Sharp curves. No brakes on empties. Return trip at 5-mile trot.

B. Long haul. Gathering mules hauled two cars to gathering station 1,000 feet from shaft over fair tracks; maximum grade against load, 1 percent. Eight-car train hauled from station to shaft. Too hard on mules; changed to gasoline haulage.

C. Good conditions. Level track, good curves. Large mules hauled six-car trains to shaft. Slow speed, 3½ miles an hour.

D. Good conditions. Track 1.5 percent in favor of load; straight track, electrically lighted. Cars run down by own weight after starting; mule trots at 5 miles. Train of four cars had to be spragged at end of run. Limit on return trip, four empty cars. Two mules haul 256 tons.

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[10] van Barneveld, Charles E., Mechanical Underground Loading in Metal Mines: Univ. of Missouri School of Mines and Metallurgy Bull., vol. 7, No. 3, May 1924, pp. 38–40.

[11] Boericke, W. F., Mule Haulage in Metal Mines: Eng. and Min. Jour., vol. 112, No. 22, Nov. 26, 1921, pp. 853–855.

PERFORMANCE [12]

A

Average tons trammed per shift		180
Average tram distance	feet	1,450
Number of mules		4
Tons of ore per mule		45
Miles traveled per mule		12.5
Ton-miles per mule		12.5
Cost per ton	cents	7.9
Cost per ton-mile	do	30

B

Average tons trammed per shift		288
Average tram distance	feet	2,613
Number of mules		9
Tons of ore per mule		32
Miles covered per mule		15.8
Ton-miles per mule		15.8
Cost per ton	cents	10.9
Cost per ton-mile	do	22.4

C

Average tons trammed per shift		400
Average tram distance	feet	2,006
Number of mules		3
Miles traveled per mule		8.5
Ton-miles per mule		26
Tons of ore per mule		64
Cost per ton	cents	5.2
Cost per ton-mile	do	14.0

D

Average tons trammed per shift		256
Tram distance	feet	1,483
Number of mules		2
Tons of ore per mule		128
Miles traveled per mule		18.1
Ton-miles per mule		36.3
Cost per ton	cents	2.8
Cost per ton-mile	do	10.1

MECHANICAL HAULAGE

In considering mechanical vs. hand or animal haulage in a mine or section of a mine, a factor that always must be considered is the duty to be performed. If there are no other mitigating conditions, it is a good general rule that for a locomotive (or any type of machine) to pay for itself it must be kept busy; in other words, idle machines pay no dividends. When mechanical haulage units can be kept busy they will usually show a saving in operating costs as compared to hand or animal tramming.

Although a single example cannot reasonably be taken as a basis for broad generalization, the following,[13] where the conditions were admittedly rather exceptional, is cited as being of interest in this connection:

> The output was 400 tons per day on two shifts, 300 tons of which came from three or four loading chutes. The maximum haul was 750 feet and the average 500 feet. Although the track was good and the grade favorable, 8 men on each of two shifts, 16 in all, using 1-ton cars, were required to load, tram, and dump the ore, thus averaging 25 tons per man-shift. A storage-battery locomotive and 3-ton Granby-type cars were placed in service, the loading chutes were widened, and ore-storage capacity was provided, after which three men could handle 300 tons from the chutes together with some ore from development headings in 2½ hours, or an average of more than 100 tons per man in one-third of a shift. Hand-tramming costs were 22 cents and locomotive-haulage costs 5.2 cents per ton, including labor and power but not maintenance or depreciation.

Mechanical haulage in metal mines is of four general types: (1) Locomotive, (2) scraper or slusher, (3) rope, and (4) belt conveyor.

Locomotives for underground use are powered by electric, compressed-air, or, rarely, gasoline motors. Diesel locomotives have been used extensively in Europe, but have not been adopted in the United

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[12] van Barneveld, Charles E., Work cited in footnote 10 (p. 171).

[13] Jackson, Chas. F., Some Notes on Underground Transportation in Metal Mines: Bureau of Mines Inf. Circ. 6326, 1930, pp. 8–9.

States. Studies are now in progress on how to adapt American Diesel locomotives to underground use and to determine conditions under which they may be employed with safety. Gasoline motors should never be employed for underground work because of the hazard from exhaust fumes.

Electric locomotives are either of the trolley or storage-battery type, and each has its particular field. For long hauls and heavy trains the trolley type usually has a decided advantage, whereas for shorter hauls, such as in gathering service from a number of loading points to a main haulageway, the storage-battery type may be preferred. Small storage-battery locomotives of the "trammer" type that can be run on the mine cage and transferred quickly from one level to another have found wide favor where the mine output is small and comes from scattered areas and several horizons in the mine. For heavy duty, storage-battery locomotives have insufficient ampere-hour capacity for long service without frequent recharging, and spare batteries may have to be supplied for each locomotive. Where haulage is confined to one shift, the batteries may be charged on the opposite shift. When charging can be done between shifts or at times when the total mine load is light, storage-battery locomotives may have an advantage over trolley locomotives in keeping down power peaks.

Cable-reel trolley locomotives are sometimes used for gathering in short headings tributary to the main haulageways, where it does not pay to install trolley wire.

Compressed-air locomotives have not been employed widely in metal mines but have certain obvious advantages in gassy coal mines. Their over-all efficiency, including that of the compressor unit, is low, and the cost of power for their operation is high. In some instances they have been installed in preference to trolley locomotives because of the hazard of exposed trolley wires, particularly at loading chutes. They receive air at high pressures (600 to 850 pounds or more), which necessitates the installation of high-pressure compressors, receivers, air mains, and charging equipment at high initial cost and requires careful attention to prevent leakages and attendant loss of power.

Although the scraper as employed underground is ordinarily considered to be a mucking and loading machine, it is also a conveying machine and in some instances is used to transport ore and rock several hundred feet. It is then an auxiliary part of the transport system for moving ore from the face to a chute or mine cars or for transferring ore from a series of stope raises, through a sublevel drift, to an ore pass.

The term "rope haulage" is used here to mean the use of hoists for pulling cars on grades or flat slopes where the grade is too steep for locomotives or the method in general use in the mine.

Conveyor systems have come into common use in coal mines during the past decade. As this is written, belt conveyors are beginning to make their appearance in metal mines as a part of the underground transportation system, and an increasing number probably will be installed during the next few years.

LOCOMOTIVE HAULAGE

It is not within the scope of this bulletin to discuss the mechanical design and construction of locomotives or other types of mine equipment; these are described in the catalogs and handbooks of the manufacturers. However, the broad question of application of different types of equipment to different mining conditions and uses, the results obtained, and costs of operation are within the province of the mining engineer.

In selecting a locomotive, its weight and horsepower must be proportioned to suit load, track grades and curvatures, condition of tracks, and type of bearings employed on the cars.

According to van Barneveld,[14] the draw-bar pull of a locomotive is usually taken as 25 percent of its weight, and the trailing load it will pull is 15 or 16 times its weight when on level, straight track. When the train is running on an adverse grade, the draw-bar pull and consequently the load that can be pulled are reduced sharply, as shown in table 26. When roller-bearing cars or cars equipped with railroad-type journals are used, and with well laid and maintained tracks, a grade of about 0.25 percent in favor of the loads usually is advocated for motor haulage, although in many mines grades of 0.50 percent are standard. Under less ideal conditions, grades of 0.5 percent may be found desirable. Table 27 gives data on actual performance of mine locomotives.

The proper weight of rail to use will depend on a number of factors, including the weight of locomotives, cars, and loads, speed of operation, and length of time the haulageway is expected to remain in service. From the standpoint of operating cost and maintenance, the heavier rails are preferred, although the first cost obviously will be greater. Light locomotives weighing 1½ or 2 tons sometimes are run on 16-pound track. Thirty-pound rail was standard in many mines for haulage with 3½-ton locomotives; with larger equipment, 40-, 50-, or 60-pound rail is employed quite commonly. Table 28 presents track data taken from actual practice and covers hand-tramming, mule-haulage, and motor-haulage operations.

TABLE 26.—*Locomotive weights, trainloads, and grades*

Grade, percent	Weight of locomotive, tons	1½	2	2½	3	3½	4	4½	5	6	7	8	9	10
0	Draw-bar pull, in pounds, on level	750	1,000	1,250	1,500	1,750	2,000	2,250	2,500	3,000	3,500	4,000	4,500	5,000
0	Gross trainload...tons	25.0	33.3	41.6	50.0	58.3	66.6	75.0	83.3	100.0	117.0	133.0	150.0	167.0
	Trailing load......do	23.5	31.3	39.1	47.0	54.8	62.6	70.5	78.3	94.0	110.0	125.0	141.0	157.0
1	Gross trainload....do	15.0	20.0	25.0	30.0	35.0	40.0	45.0	50.0	60.0	70.0	80.0	90.0	100.0
	Trailing load......do	13.5	18.0	22.5	27.0	31.5	36.0	40.5	45.0	54.0	63.0	72.0	81.0	90.0
2	Gross trainload....do	10.7	14.3	17.9	21.4	25.0	28.6	32.1	35.7	42.8	50.0	57.1	64.3	71.4
	Trailing load......do	9.2	12.3	15.4	18.4	21.5	24.6	27.6	30.7	36.8	43.0	49.1	55.3	61.4
3	Gross trainload....do	8.3	11.1	13.9	16.7	19.5	22.2	25.0	27.8	33.3	38.9	44.4	50.0	55.5
	Trailing load......do	6.8	9.1	11.4	13.7	16.0	18.2	20.5	22.8	27.3	31.9	36.4	41.0	45.5
4	Gross trainload....do	6.8	9.1	11.4	13.7	16.0	18.2	20.5	22.8	27.3	31.9	36.4	41.0	45.5
	Trailing load......do	5.3	7.1	8.9	10.6	12.4	14.2	16.0	17.7	21.3	24.9	28.4	32.0	35.5
5	Gross trainload....do	5.7	7.7	9.6	11.5	13.5	15.4	17.3	19.2	23.1	26.9	30.8	34.6	38.5
	Trailing load......do	4.2	5.7	7.1	8.5	10.0	11.4	12.8	14.2	17.1	19.9	22.8	35.6	28.5
6	Gross trainload....do	5.0	6.6	8.3	10.0	11.6	13.3	15.0	16.6	20.0	23.3	26.6	30.0	33.3
	Trailing load......do	3.5	4.6	5.8	7.0	8.1	9.3	10.5	11.6	14.0	16.3	18.6	21.0	23.3
7	Gross trainload....do	4.4	5.9	7.3	8.8	10.3	11.7	13.2	14.7	17.6	20.6	23.5	26.5	29.4
	Trailing load......do	2.9	3.9	4.8	5.8	6.7	7.7	8.7	9.7	11.6	13.6	15.5	17.5	19.5
8	Gross trainload....do	4.0	5.2	6.6	7.9	9.2	10.5	11.8	13.1	15.8	18.5	21.0	23.7	26.3
	Trailing load......do	2.5	3.2	4.1	4.9	5.7	6.5	7.3	8.1	9.8	11.5	13.0	14.7	16.3
9	Gross trainload....do	3.6	4.8	6.0	7.1	8.3	9.5	10.7	1.9	14.3	16.7	19.0	21.4	23.8
	Trailing load......do	2.1	2.8	3.5	4.1	4.8	5.5	6.2	6.9	8.3	9.7	11.0	12.4	13.8
10	Gross trainload....do	3.2	4.3	5.4	6.5	7.6	8.7	9.8	10.9	13.0	15.2	17.4	19.5	21.7
	Trailing load......do	1.7	2.3	2.9	3.5	4.1	4.7	5.3	5.9	7.0	8.2	9.4	10.5	11.7

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[14] van Barneveld, Charles E., Work cited in footnote 10 (p. 171), p. 60.

TABLE 27.—*Data on actual performance of mine locomotives*

Reference, Bureau of Mines information circular or other publication	Weight, tons, and type of locomotive	Rated draw-bar pull, in pounds, and full-load speed, miles per hour	Motors and batteries	Cars			Average haul, feet	Ore hauled per locomotive shift, tons
				Weight, pounds	Load, pounds	Number in train		
6107 [1]	2 6-ton in tandem trolley				16,000	15	4,000	[2] 1,300
		6 miles		7,000	12,000	20		
Carmichael [3]	3 6-ton in tandem [4]	10,000 starting pull	2 30-h. p. each locomotive	9,700	20,000	30	3,300	[1] 1,800
6138 [5]	6.5-ton trolley		2 250-volt motors, Edison, A cells.		11,000	5 or 6		
6138 [5]	1.75 and 4-ton battery				5,000	6–8	1,200	100–150
6145 [6]	1½-ton battery				2,000	6–8	1,200	100–150
6160 [7]	4-ton trolley		2 motors		2,000 to 4,000	8–14		
6167 [8]	8-ton compressed air	3,000 pounds			9,200	20–25	3,000	
6168 [9]	4-ton trolley	2,000 pounds, 5.0 miles	2 20-hp. motors		7,200		2,300	800–1,000
6168 [9]	do	do			4,000	10	1,700	300–400
6169 [10]	8-ton trolley		2 motors		10,000	10–25	6,600	
6170 [11]	15-ton trolley	6,500 pounds, 15 miles	2 90-hp., 275-volt motors		5,000	30 max., 14 on 2¾ % adverse grade for 4,800 feet	4,170	
6180 [12]	6.5-ton trolley	3,250 pounds	2 39½-hp. 250-volt motors		4,000	10		
6186 [13]	18-ton trolley				20,000	44	10,000	
6238 [14]	6.5-ton trolley			4,000	8,400	16	10,000	
6238 [14]	4.5-ton battery		78 cells		3,000	10		
6240 [15]	10-ton battery				5,000	8–10	2,000	
6247 [16]	10-ton trolley		2 motors	5,300	7,000	6–10	4,200	[17] 175
6247 [16]	do		do	5,300	7,000	12	3,200	1,000
6247 [16]	4-ton trolley and cable reel			4,000	7,000 to 8,000	6–10		
6247 [16]	1.5-ton battery				2,000	5–6	400	65–100
6260 [18]	3-ton battery				2,000	10		240
6260 [18]	7-ton battery				12,000	10		
6290 [19]	4-ton, 2 in tandem	2,000 pounds	2 240-volt motors	3,450	9,000	20	10,000	[1] 650±
6371 [20]	4-ton trolley	2,600 pounds, 4.7 miles	2 motors	1,100	2,000	35	10,560	About 250
	3-ton trolley	1,200 pounds, 5.2 miles	do					
6440 [21]	25-ton trolley	7 miles	2 75-hp. motors		80,000	9	9,500	1,250
6512 [22]	1.5-ton battery	600 pounds, 4.5 miles			2,000	6	1,800	75
6565 [23]	10-ton trolley	8 miles			10,000	15 max		

See footnotes at end of table.

TABLE 27.—*Data on actual performance of mine locomotives*—Continued

Reference, Bureau of Mines information circular or other publication	Weight, tons, and type of locomotive	Rated draw-bar pull, in pounds, and full-load speed, miles per hour	Motors and batteries	Cars			Average haul, feet	Ore hauled per locomotive shift, tons
				Weight, pounds	Load, pounds	Number in train		
6565 [23]	25-ton trolley	9,000 pounds, 7 miles			47,600	12	11,600 max.	4,000±
6586 [24]	1.5-ton battery	600 pounds, 4.5 miles			2,000	5–6	350	
6661 [25]	4.5-ton trolley				7,000	10–12		580
6661 [25]	4-ton battery		84-cell Edison, 300 amp.-hr		3,500	8–12		
Bull. 306 [26]	13-ton trolley	6,500 pounds			14,000	5		
Bull. 306 [26]	3-ton trolley and battery	1,000 pounds, 6 to 8 miles	Battery, 48 cells		4,000 to 10,000	2, 4, 6	Over 1,000	

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[2] Per tandem unit.

[3] Carmichael, Norman, and Kiddie, John, Development of Mine Transportation in Clifton-Morenci District: Trans. Am. Inst. Min. and Met. Eng., vol. 70, 1924, pp. 826–855.

[4] 1 controller for 3 locomotives.

[5] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[6] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

[7] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

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[9] Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Bureau of Mines Inf. Circ. 6168, 1929, 32 pp.

[10] Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

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[12] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Michigan: Bureau of Mines Inf. Circ. 6180, 1929, 10 pp.

[13] Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alasks: Bureau of Mines Inf. Circ. 6186, 1929, 18 pp.

[14] Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6238, 1930, 13 pp.

[15] Gardner, E. D., Mining Practice at Harmony Mines Co., Baker, Idaho: Bureau of Mines Inf. Circ. 6240, 1930, 8 pp.

[16] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Bureau of Mines Inf. Circ. 6247, 1930, 41 pp.

[17] Plus timber and supplies.

[18] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[19] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Bureau of Mines Inf. Circ. 6290, 1930, 18 pp.

[20] Dailey, M. J., Mining Methods and Costs of the Silver King Coalition Mines Co., Park City, Utah: Bureau of Mines Inf. Circ. 6371, 1930, 12 pp.

[21] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[22] Spiers, James, Mining Methods and Costs at Central Eureka Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6512, 1931, 12 pp.

[23] Webb, J. S., and Skinner, T. W., Mining Methods and Costs at the Braden Copper Co.'s Mines, Sewell, Chile: Bureau of Mines Inf. Circ. 6565, 1932, 13 pp.

[24] Knaebel, J. B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N. Y.: Bureau of Mines Inf. Circ. 6586, 1932, 35 pp.

[25] Livingston, A., Mining Methods and Costs at Fresnillo, Zacatecas, Mexico: Bureau of Mines Inf. Circ. 6661, 1932, 31 pp.

[26] Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bureau of Mines Bull. 306, 1929, 192 pp.

TABLE 28.—*Track data*

Reference, Information Circular No.	Type of haulage	Capacity of cars	Weight of locomotive	Weight of rail, pounds	Ties		Grade of tracks, percent	Curve radius, feet	Track gage, inches
					Size	Spacing			
6138 [1]	Motor	2.5 tons; 5.5 tons	1½, 4, and 6½ tons.	40					30
6145 [2]	do	20 cu. ft	"Trammer" size	18 and 24			0. 5		18
6149 [3]	do	2 to 7 tons	1 to 6 tons	40					24
6150 [4]	Mule	0.6 to 0.7 ton		12	3″ by 4″				18
6150 [4]	Hand			8					
6159 [5]	Mule and hand	1.5 tons		20					24
6160 [6]	Motor	1 and 2 tons	4 tons	30			4 max.		24
6167 [7]	do	4.6 tons	8 tons	45 and 35	6″ by 8″ by 4′6″		0. 25–0. 33	50 to 60	30
6168 [8]	do	2 to 3.6 tons	3 to 4.5 tons	30	6″ by 6″ by 3′0″		0. 5	50	18
6169 [9]	do	2 and 5 tons	8 tons	40 and 60				60 standard, 35 minimum.	30
6170 [10]	do	2.5 tons	15 tons	30–56			0. 5		24
6170 [10]	Mule	2.5 tons		20					
6179 [11]	Motor	2 to 3.5 tons	6.5 tons	40					30
6180 [12]	do	2 tons	6.5 tons	40					30
6186 [13]	do	10 tons	18 tons	50					30
6237 [14]	do	20 cu. ft	3 and 4 tons	30			0. 5	40	18
6237 [14]	Hand	20 cu. ft		12, 24				15	18
6240 [15]	Motor	2.5 tons	10 tons	30	4″ by 6″ by 3′6″		0. 5		18
6240 [15]	Hand	0.75 ton		12					
6247 [16]	Motor	3.5 to 4 tons	1.5 to 4 tons	30	6″ by 8″ by 4′0″	24″	0. 5	35 min	18
6247 [16]	do	7 tons	10 tons	30	6″ by 8″ by 6′0″	24″	0. 5	60 min	36
6247 [16]	Hand	1 ton		16	4″ by 6″ by 4′0″	24″			18
6250 [17]	Motor	2 tons	5 tons	30				25 min	
6250 [17]	Hand	16 cu. ft							
6260 [18]	Motor	1 ton	3 tons	20	4″ by 8″ by 3′0″		0. 25		18
6260 [18]	do	6 tons	7 tons	30	5″ by 8″ by 4′0″		0. 25		24
6290 [19]	do	3.5 and 4.5 tons	4 tons	40, 45	4″ by 6″ by 3′0″	30″	0. 25–0. 50	30 min	18
6290 [19]	Hand	16 cu. ft		16					
6294 [20]	do	22 cu. ft		16			0. 5		18
6311 [21]	do	1 ton		16					18
6325 [22]	do	3 tons		30					
6325 [22]	Motor	3 tons	6 tons						
6327 [23]	Hand	18 cu. ft		20	4″ by 6″ by 30″	30″	1. 5 max.		18
6327 [23]	Motor	50 cu. ft., 3 tons	3½ tons	30	4″ by 6″		0. 5		24
6348 [24]	do		2, 4, 6 tons	50, 40, 30			0. 5		24
6361 [25]	do	60 cu. ft., 5 tons	5 tons	35	5″ by 5″ by 5′	24″	0. 5	100′54″	36
6368 [26]	do	40 and 18 cu. ft		16					24
6369 [27]	do	40 cu. ft., 2¼ tons	2, 4, 6 tons	40	5″ by 5″	18″	0. 5		24

See footnotes at end of table.

TABLE 28.—*Track data*—Continued

Reference, Information Circular No.	Type of haulage	Capacity of cars	Weight of locomotive	Weight of rail, pounds	Ties		Grade of tracks, percent	Curve radius, feet	Track gage, inches
					Size	Spacing			
6371 [28]	do	17 cu. ft	4 tons 3 tons	30–20 16	4″ by 6″ by 32″	24″	0. 5		18
6377 [29]	Hand	14.5 cu. ft		16	3″ by 6″				18
6380 [30]	Motor	4 tons, 65 cu. ft	6 and 10 tons	40					30
6390 [31]	do	4 tons	6 tons	40					30
6402 [32]	Horse	1.8 tons		18, 24					18
6407 [33]	Motor	3 tons, 2 tons	8 tons "trammer."						24
6440 [34]	do	40 tons, 25 tons	25 tons	75			0. 5		56½
6440 [34]	do	30 cu. ft		40	6″ by 6″ by 30″	24″		50 min	18
6440 [34]	Hand	18 cu. ft		16	4″ by 6″				18
6462 [35]	Hand and motor	17 cu. ft., 50 cu. ft	3 tons	16 and 12			0. 5		18
6480 [36]	Motor	40 cu. ft	2½ tons	20	Steel	36″	0. 3		23
6512 [37]	do	16 cu. ft., 1 ton	1½ tons	25	4″ by 6″ by 30″		0. 4	15	18
6526 [38]	do	7.5 tons	13 tons, 10 tons	50					52
6565 [39]	do	23.8 tons	25 tons	60			0. 5		30
6601 [40]	do	4.2 tons	4 to 6.5 tons	32			0. 25	40	26
6661 [41]	do	70 cu. ft	4.5 tons	25	5″ by 6″ by 42″	24″	0. 5		20
6804 [42]	do	21 and 57 cu. ft	1.5 tons	20 and 30	4″ by 6″ by 30″				18
6815 [43]	do	17 tons	18 and 15 tons	80			0. 5		36
6815 [43]	do	20 tons	20 and 40 tons	85			0. 5		36
6815 [43]	do	100 cu. ft	6 tons	45			0. 5		24

[1] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[2] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

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[4] Banks, L. M., Mining Methods and Costs in the Waco (Mo.) District: Bureau of Mines Inf. Circ. 6150, 1929, 10 pp.

[5] Keener, O. W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

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[8] Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Bureau of Mines Inf. Circ. 6168, 1929, 32 pp.

[9] Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

[10] Jackson, Chas. F., Method of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Bureau of Mines Inf. Circ. 6170, 1929, 21 pp.

[11] Eaton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6179, 1929, 15 pp.

[12] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.: Bureau of Mines Inf. Circ. 6180, 1929, 10 pp.

[13] Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Bureau of Mines Inf. Circ. 6186, 1929, 18 pp.

[14] Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Bureau of Mines Inf. Circ. 6237, 1930, 21 pp.

[15] Gardner, E. D., Mining Practice at Harmony Mines Co., Baker, Idaho: Bureau of Mines Inf. Circ. 6240, 1930, 8 pp.

[16] Catron, William, Mining Methods and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Bureau of Mines Inf. Circ. 6247, 1930, 41 pp.

[17] D'Arcy, R. L., Mining Practice and Methods at the United Verde Extension Mining Co., Jerome, Ariz.: Bureau of Mines Inf. Circ. 6250, 1930, 11 pp.

[18] Nelson, W. I., Mining Methods and Costs at the Plumas Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[19] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Bureau of Mines Inf. Circ. 6290, 1930, 18 pp.

[20] Reeder, E. C., Method and Cost of Mining Fluorspar at Rosiclare, Ill.: Bureau of Mines Inf. Circ. 6294, 1930, 10 pp.

[21] Vanderburg, W. O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6311, 1930, 14 pp.

[22] Haselton, W. D., Underground Mining Practice and Costs at a Mesabi Range (Minn.) Mine Using the Top-Slicing System: Bureau of Mines Inf. Circ. 6325, 1930, 11 pp.

[23] Hezzelwood, G. W., Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.: Bureau of Mines Inf. Circ. 6327, 1930, 15 pp.

[24] Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.: Bureau of Mines Inf. Circ. 6348, 1930, 13 pp.

[25] Kniffin, L. M., Mining and Engineering Methods and Costs of the Hanover Bessemer Iron & Copper Co., Fierro, N. Mex.: Bureau of Mines Inf. Circ. 6361, 1930, 20 pp.

[26] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[27] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

[28] Dailey, M. J., Mining Methods and Costs at the Silver King Coalition Mines Co., Park City, Utah: Bureau of Mines Inf. Circ. 6371, 1930, 12 pp.

[29] Richard, F. A., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Bureau of Mines Inf. Circ. 6377, 1930, 13 pp.

[30] Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 5 of the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6380, 1930, 10 pp.

[31] Graff, W. W., Mining Practices, Methods, and Costs, Mine No. 4 of the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6390, 1930, 8 pp.

[32] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Bureau of Mines Inf. Circ. 6402, 1931, 11 pp.

[33] Brown, U. E., Mining Methods and Costs of the Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho: Bureau of Mines Inf. Circ. 6407, 1931, 9 pp.

[34] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[35] Moorehead, W. R., Methods and Costs of Mining Quicksilver at the New Idria Mine, San Benito County, Calif.: Bureau of Mines Inf. Circ. 6462, 1931, 14 pp.

[36] Savage, J. P., Tunnel-Driving Methods Used at the Ojuela Unit of the Compañia Minera de Peñoles, S. A., Ojuela, Durango, Mexico: Bureau of Mines Inf. Circ. 6480, 1931, 8 pp.

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[38] Vivian, Harry, Deep Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.: Bureau of Mines Inf. Circ. 6526, 1931, 20 pp.

[39] Webb, J. S., and Skinner, T. W., Mining Methods and Costs of the Braden Copper Co.'s Mines, Sewell, Chile: Bureau of Mines Inf. Circ. 6565, 1932, 13 pp.

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[41] Livingston, A., Mining Methods and Costs at Fresnillo, Zacatecas, Mexico: Bureau of Mines Inf. Circ. 6661, 1932, 31 pp.

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[43] Brennan, C. V., The Mining Operation at the Property of the Britannia Mining & Smelting Co., Ltd., Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6815, 1934, 36 pp.

Tables 27 and 28 indicate the range in capacities of ore cars commonly employed in metal mines. For large output and long hauls the use of large-capacity cars usually is justified from the standpoint of economy in operation. Other considerations may be more important, however, and small cars may be employed for a number of reasons. Thus, the size of drifts it is feasible to drive and maintain often limits the size of car that can be used. In other instances ore may be gathered from many scattered points in the mine and trammed by hand to the main motor haulageway, and it may be cheaper to haul the small hand tram-cars in trains even over long distances than to dump them and reload into cars of larger capacity for locomotive haulage. Scrapping of existing small-capacity equipment and substitution of equipment of larger capacity often may be desirable after a mine has advanced from small- to large-scale operation, but in other instances, where the earlier equipment is still in good condition, the capital outlay required to replace it may not be justified by the anticipated savings in direct haulage costs.

Much attention has been given to details of design and construction of mine cars, but a discussion of this subject would be too voluminous for inclusion in this bulletin.

Today most mines employ steel cars equipped with roller bearings, though in a few large mines railroad-type journal bearings are used. The principal types of cars are distinguished chiefly by the methods of dumping, which, in turn, largely determine the several characteristic designs. End-dump and side-dump cars with end or side doors that are tipped endwise or sidewise to the dumping position have been mentioned under "Hand Tramming," and these types, especially the side-dump cars, are used with motor haulage also.

Side-dump cars of the rocker type have been popular in recent years. The car body is V-shaped in cross section, the bottom of the V being rounded. Rounded trunnions at each end of the body below the center of gravity rock or roll on a frame attached rigidly to the truck frame; the body is rocked over on its side to discharge the contents of the car. This type of car can be emptied while it is moving slowly over the dumping point.

The Granby car is a favorite type for use in connection with locomotive haulage where rapid dumping of trains with minimum labor is desired. It is designed to be dumped while in motion. A roller on the under side of the car body rides up on a ramp constructed along one side of the track at the dumping point and lifts that side of the body. As the body tips, a side door on the opposite or low side is raised, and the contents is discharged. The locomotive pushes the train slowly along the ramp, and each car is dumped in turn without having to stop the train. The cars also can be dumped by means of an air jack or by a rope and winch at points where no dumping ramp is provided. If there is ample pocket capacity the train can be dumped and returned immediately for its next load in less time than ordinarily would be required to switch out the loaded train and couple up with a standing empty one. The train crew does the dumping, and a minimum number of cars is required as no cars are idle at any time. The Granby car is heavy, and some operators claim it is difficult to keep clean if the ore is at all sticky, but the advantage of quick dumping and low operating cost may often outweigh these objections.

The gable-bottom car has been employed widely, but in recent years the trend has been toward rocker-bottom or other types of cars. Gable-bottom cars have a swinging door on each side of the car, hinged at the top, which, when unlatched, permits the contents to discharge. The gable-bottom construction reduces the capacity of the body, and the doors usually require considerable repair work to keep them tight and prevent leakage of fine ore along the haulage roads. Bottom-discharge cars are used at some mines but are not as common as other types.

Solid-body cars with the bodies attached rigidly to the truck frame have a number of advantages but require a cradle or rotary dump for discharging and for this reason are not well-adapted for handling and distributing ore or waste to a number of dumping points. Where ore only is handled and there is but one dump point or large tonnages are dumped at each point, the cost of installing the dumping mechanism may be outweighed by the advantages of this type of car. They are low in first cost in relation to their capacity, they are sturdy and have no doors to maintain and keep in repair, no leakage of fines occurs as in the case of cars with poorly-fitting doors, and the height above the rail is low in proportion to their capacity.

Each type of car mentioned has its advantages under certain conditions determined by such factors as tonnage to be handled and the number of loading and dumping points; size of haulageways; physical character of the material handled, whether fine or coarse, wet, dry, or sticky, and the method of loading. Thus, a low car is desirable for loading by hand shoveling or by power scraper in drifts and crosscuts. Likewise, the height of loading chutes above the rail may limit the height of the car. If cars are to be hoisted on cages and dumped on the surface the size of the hoisting compartments and cages will limit the size of car that can be employed. For handling fine material such as sand filling as well as fine ore, a solid-body car without doors has obvious advantages.

TRAMMING AND HAULAGE COSTS

Costs of tramming and haulage are available from only a few mines, and it is difficult to make accurate comparisons because of differences in accounting practices at different mines. It is often the practice to combine costs of hand mucking and tramming, especially in development work. Most operators charge loading from chutes against tramming cost, some include also cost of chute maintenance and repairs, and some combine tramming and hoisting under the general head of transportation. In some instances dumping cars at the skip pocket may be charged to tramming if the dumping is performed by the train crew, whereas in others, where dumping is done by a separate station attendant or shaft crew, this operation may be charged to hoisting. Some mines segregate waste tramming from ore tramming, whereas many others do not.

A few examples of tramming and haulage costs are given in table 29, which, for the reasons above stated, cannot be compared precisely.

TABLE 29.—*Examples of underground tramming and haulage costs*

Mine and year	Type of haulage	Approximate average distance, feet	Capacity of cars	Costs per ton of ore			
				Labor	Power	Supplies, shop costs, miscellaneous	Total
Eighty-Five, N. Mex.,[1] 1930.	Hand tramming	1, 200	16 cu. ft	$0. 59		$0. 08	$0. 67
Elkoro, Nev., 1930	Hand, mule, and battery locomotive.	[2] 1, 600	20 and 26 cu. ft	. 295	$0. 010	. 044	. 349
Cresson, Colo., 1931	Hand	600	16 cu. ft				. 63
Cresson, Colo., 1932	do	600	do				. 52
Hog Mountain, Ala., 1935–36.	Hand. Cars caged	350	16 and 20 cu. ft	. 189		. 047	[3]. 236
Alaska-Juneau, Alaska., 1928.	Trolley locomotive	10, 000	10 tons	. 0811	. 0035	. 0290	. 1136
Mineville, N. Y., 1927.	Hand and trolley locomotives.	1, 030	1½ tons; 3½ tons.	. 390	. 013	. 065	. 468
Waco, Mo., 1927–1928.	Hand		1,200-lb. cans	. 057		. 007	. 064
No. 8, Southeast Missouri, 1928.	Trolley locomotive		1 and 2 tons	. 1073	. 0138	. 0565	. 1776
El Potosi, Mexico, August 1933.	Battery locomotive		21 cu. ft. and 57 cu. ft.	. 010	. 002	. 008	. 020
United Verde Extension, Ariz., 1928.	Trolley locomotive, small battery locomotive, hand.		30 cu. ft.; 16 cu. ft.				. 357
Eureka Standard, Utah, Feb. 1933.	Battery locomotive		0.9 ton	[4]. 17			. 17
Montreal No. 4, Wis., 1928.	Trolley locomotive		40 cu. ft	. 121	[5]. 009	. 025	. 155
Montreal Nos. 5 and 6, Wis., 1928.	do		do	. 095	. 010	. 015	. 120
Ray, Ariz., 1928	Compressed-air locomotive.	3, 000	103 cu. ft	. 0585	. 0078	. 0246	. 0909
Braden, Chile, 1928	Hand			. 0393		. 0027	. 0420
Do	Trolley	11, 000	23.8 tons	. 0100	. 0010	. 0056	. 0166
Britannia, B. C.:							
East Bluff	Trolley locomotives		13 to 20 tons	. 031	. 020	. 009	. 060
Victoria	Hand and trolley locomotives.		100 cu. ft	. 134			. 134
Fairview (1933–1934).	Trolley locomotives		13 to 20 tons	. 046	. 030	. 001	. 077
Miami, Ariz., 1925–1929.	do	5, 000	86 cu. ft				. 05314

[1] Cost includes tramming ore and waste, chute-blasting, and supplies such as mine cars, oil, etc., but not track, which is $0.16 additional.
[2] Locomotive.
[3] Includes some mucking.
[4] Includes hauling waste.
[5] Includes power cost of scraping.

SCRAPING

Scraping has been discussed in detail in an earlier Bureau of Mines publication.[15] The scraper was first employed in metal mines primarily as a mucking and loading machine but has come to be used widely for transporting ore and rock for limited distances. Thus, scrapers are used in stopes to drag ore from the face to the chutes, to place waste filling in the stopes, and, in scraper drifts and crosscuts, to drag ore from a line of raises to a common main ore pass or directly into cars on the haulage level. Direct loading into cars in headings and flat stopes by means of a scraper slide has been discussed briefly in the section on "Mine Development."

In considering scraper transportation of ore, the digging function of the scraper cannot be disregarded, since it must first fill itself by digging before it can transport its load by dragging. A scraper designed for maximum digging efficiency may not work well as a means of transportation; it may catch on rough spots and spill most of its load along the way, may dig into the bottom if it is soft, or tear up flooring laid to protect the bottom. These difficulties can be overcome by proper design, though sometimes at the sacrifice of some digging efficiency.

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[15] Jackson, Chas. F. Underground Scraping Practice in Metal Mines: Bureau of Mines Manuscript Rept. 1, 1933, 88 pp.

Two general types of scrapers are employed underground—the box type and the hoe type. Both are bottomless, although some are constructed with a back plate curved forward to meet the floor at a flat angle, forming a partial sloping bottom. Figure 25 shows one form of hoe scraper at *A*. The box scraper differs from this principally in that it has sides. (Fig. 68.) Figure 25, *B*, shows a semihoe scraper that has short side plates extending upward and forward from the digging edge to the bale.

In general, the box scraper is applicable for handling finely broken or granular material, whereas the hoe type is better adapted to hard, chunky ore. The box scraper will dig fine material and, by reason of the sides, retain the load better; but in digging blocky ore the sides tend to ride up on the lumps and thus prevent penetration of the cutting edge or teeth into the muck pile, especially if the scraper is light in weight. By allowing spill to accumulate along the haulageway it will form banks on each side of the scraper rut, which will prevent further spillage even when a hoe scraper is used.

The proper size (capacity) of scraper for a given installation will depend on the quantity of ore to be handled and the rate of handling desired and will be limited in some instances by the dimensions of the heading in which it is to operate. The weight of the scraper for any particular condition is influenced largely by the size and weight of the broken ore. In soft, dry, loose muck a light scraper usually will dig satisfactorily, but in hard, chunky ore adequate weight is a primary requirement. Probably more initial scraper installations have failed because the operators started with a scraper too light to penetrate the muck pile or with a hoist too small to load and pull a heavy scraper than for any other reason.

As shown by graphic power-input charts taken from scrapers in operation, the maximum power during the scraping cycle is required when the scraper is digging into the pile when it is often two, three, or more times that required during the dragging period.

For transportation purposes the scraper should be designed so that its digging action will cease or be greatly reduced after the scraper is filled. In somewhat lumpy ore the sides of the box scraper usually will accomplish this. In soft, fine ore it may be necessary to curve the top part of the back plate forward, as shown in figure 68 at *E*. The force of the load against this top section then tends to lift the scraper. The same method may be employed with hoe-type scrapers, or, as an alternative, short, tapered side plates may be added (fig. 25, *B*).

Scrapers are best adapted to transportation underground where large quantities of ore must be moved comparatively short distances. Usually scrapers of large capacity are necessary for this, which in turn requires the use of hoists larger than are customarily employed for loading in drifts and crosscuts, the nature of the material handled being the same. The trend during the past 15 years has been toward the use of larger, more powerful scraper hoists capable of pulling heavier loads at higher speeds. Overpowered hoists are desirable for

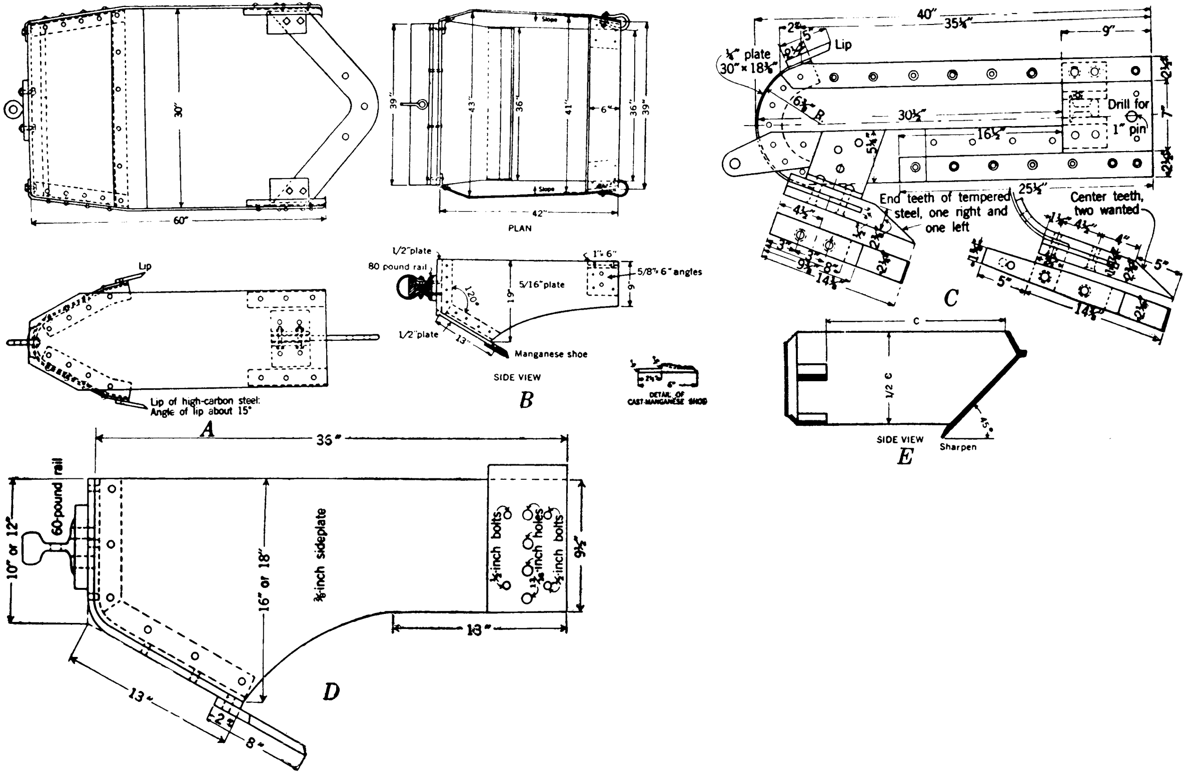
60"
30"
Lip
Lip of high-carbon steel: Angle of lip about 15°
A
39"
43"
36"
41"
6"
36"
39"
Slope
Slope
42"
PLAN
1/2"plate
80 pound rail
1"× 6"
5/16"plate
5/8"× 6" angles
19"
9"
120°
1/2"plate
13"
Manganese shoe
SIDE VIEW
B
DETAIL OF CAST-MANGANESE SHOE
40"
35½"
9"
2"
5"
Lip
½" plate 30"× 18½"
6½" R
30½"
16½"
Drill for 1" pin
2½"
7"
2½"
5½"
End teeth of tempered steel, one right and one left
25½"
Center teeth, two wanted
4½"
3"
8"
9½"
14½"
2½"
4½"
4"
5"
5"
14½"
2½"
C
c
1/2 c
45°
SIDE VIEW
Sharpen
E
35"
10" or 12"
60-pound rail
16" or 18"
⅜-inch sideplate
½-inch bolts
1¹/₁₆-inch holes
½-inch bolts
9½"
13"
13"
2
8"
D

FIGURE 68.—Forms of box scrapers. *A*, Reversible box scraper used in soft iron ore, Negaunee, Mich. *B*, Box scrapers used in soft iron ore, Negaunee, Mich. *C*, Reversible Quincy box scraper. *D*, Semibox scraper. *E*, Box scraper with upper part of back turned forward.

scraper work, as they are subjected to rough use and frequent momentary overloading, and though the first cost is higher, the maintenance cost is considerably less.

A few years ago operators in the Lake Superior iron-ore districts were replacing all their smaller scraper hoists with hoists of 15 or more horsepower. More recently some companies are replacing 15-horsepower hoists with hoists rated at 25 horsepower. This is mentioned because scraping has been practiced to the virtual exclusion of hand shoveling and sublevel hand tramming in these districts for 15 years, and hence the experience there should indicate the most efficient practice.

Table 30 presents data on a few operations where scrapers are employed principally for transportation. With the one exception noted, all the hoists were electrically driven. The performance data include digging, as it is impossible, without making time studies, to separate the digging and transporting functions of the scrapers.

Except on short drags, slow rope speeds greatly reduce the amount of ore that can be handled. A slow speed is desirable for digging, and this may be obtained in part by slipping the clutch during the digging part of the cycle. Better practice for transfer-level work or other long drags is to employ a two-speed hoist—low speed for digging and high speed for dragging. This can be done without any difficulty in selecting a motor of the proper horsepower, as the maximum pull occurs during digging, whereas higher speed is required during dragging when the load is lighter.

The data in table 30 were obtained in 1932, and since that time considerably higher speeds have been adopted for use, 350 feet per minute being common; speeds of 450 feet for pulling the load and 650 feet for returning the empty scraper have become standard in some mines.

A smooth bottom is required when dragging is done at the higher speeds, and it often becomes necessary to lay steel rails or plank flooring for the scraper to slide on.

Scraper hoists for underground use are powered by electric or compressed-air motors. From the standpoint of power consumption, the electric motor is to be preferred. With air hoists, theoretically about 4 horsepower is required on the air compressor to deliver 1 horsepower of useful work to the hoist. In actual practice the air-operated hoist requires six or eight times as much primary power as the electrically driven hoist. Actual meter tests of electric scraper hoists in scraping several thousand tons of ore have shown power consumption as low as 0.065 kw.-hr. per ton scraped with short drags and under favorable conditions. With electric drive the power requirement is usually less than 0.5 kw.-hr. per ton, except for very long drags or exceptionally difficult conditions. Matson [16] has presented figures showing actual comparative costs of scraping with air hoists and with electric hoists. In the example given, 604,005 tons of ore were scraped with air hoists at a power cost of $0.0346 per ton and 91,124 with electric hoists at a power cost of $0.0042 per ton, a ratio of about 8 to 1.

Air hoists have certain advantages, however. Compressed air usually is available in all active sections of the mine, the exhaust

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[16] Matson, Robert C., Scraping Practice in the Michigan Iron Mines of the Lake Superior District: Michigan Coll. Min. and Tech. Bull. 4, vol. 2, 1929, 75 pp.

TABLE 30.—*Data on scraper transportation*

(Scraping distance, 100 feet or more)

Type of ore	Hoist		Details of scraper				Scraping distance, feet	Remarks
	Horsepower	Rope speed, feet per minute	Type	Width, inches	Approximate weight, pounds	Approximate load		
Soft iron ore	15	225	Box	34		10 cu. ft.	100 max.	Level pull to raises.
Do	15	240–280	do	48		18 cu. ft.	60–150	Level pull to raises. Maximum loading rate, 60 tons per hour.
Hard and soft iron ore, some chunks	15	240–280	Hoe	42		9 cu. ft.	125 max.	Pull down 30°–35° slope.
Hard, chunky rock and soft iron ore	25	230	Semihoe	54	900	18 cu. ft.	50–150	25 tons per hour mucking into large cars.
Large blocks of zinc ore in dolomite gangue.	25	200 high; 144 low	Arc-back hoe	72	2,660	2,000 lbs.	400 max., 175 av.	Pull down 20° slope, 50 tons per man-shift, including operators, helpers, and repairmen.
Coarse and fine copper-bearing conglomerate.	35 (air)		Hoe	48	1,500	13 cu. ft.	150 max.	100 tons per shift loading cars on level.
Very hard ore, large angular blocks	60		Arc-back hoe	80	2,800	10,000 lbs.	150 max.	In scraper drift, 500 tons per 8 hours.
Hard, siliceous ore in fine, sticky gangue.	15	240	Semihoe	40	650	700 lbs.	200 max.	10 tons per hour with 200-foot pull to raises.
Hard, siliceous ore in flat slabs; chunks and fines.	15	230	do	40	650	700 lbs.	100 max.	70 tons per shift, 1-man loading cars.
Iron ore; large slabs	55	130–150	Box with teeth	48	3,100	6,500 lbs.	200 max.	Average, 300 tons per shift loading into cars.
Hard, blocky magnetite	25	200	Hoe	48	1,400	2,240 lbs.	330 max.; 180 av.	27.3 tons per man-shift scraping to cars.
Large, heavy, angular blocks	150	190 return, 174 pull.	do	84	3,600	7,600 lbs.	100 max.; 75 av.	96 tons per hour on transfer level. Actual time operating, 60 percent.

has some cooling and ventilating effect, the speed of air motors may be controlled by the throttle, and if overloaded they will stall without damaging the motor.

Electric-hoist motors may be either D. C. or A. C. If direct current is employed, power usually is taken from the trolley wires. Where numerous scrapers are in use and alternating current is available at 440 or 550 volts, it usually pays to run special transmission lines and employ A. C. motors, although D. C. motors have certain advantages for scraping. Thus, speed control is obtained more easily with D. C. motors; slow speed with high torque during digging and higher speeds for dragging can be provided for in the motor windings.

In straight transfer drifts or on other straight drags it is usually necessary to employ only one man for operating the scraper provided illumination is adequate. Sometimes a helper may be necessary at the loading point if the ore is very blocky or along the haulageway if the drift is crooked.

Next to labor, general repairs and rope renewal usually are the largest items in the cost of scraper operations, whereas cost of power is low, especially if hoists are electrically driven. Ropes receive rough treatment and have a short life because of overwinding on small drums and abrasion from rubbing on the back of the scraper and against rock. Scraping costs at one mine were distributed as follows:

	Cents per ton
Labor	5. 27
Rope	1. 43
Repairs and renewals including sheaves	1. 86
Power	. 69
	9. 25

In this example the ore was in large chunks, the scrapers weighed 1,760 pounds and had a capacity of 1 ton of ore, and the average scraping distance was 175 feet, the maximum being 400 feet. Five and one-third tons of ore was handled per foot of ½-inch wire rope.

At another mine the cost of loading in drift headings with scraper and slide and 25-horsepower electric hoists was divided as follows:

	Cost per ton
Operating labor	$0. 0500
Maintenance (excluding rope) [1]	. 0572
Rope	. 0263
Power	. 0160
	. 1495

[1] Includes installation of long power line.

In top-slicing operations conducted by another company 455,000 tons of soft iron ore was scraped, 6 by 19⅝-inch rope with independent wire center being used. With hoist drums of less than 9-inch diameter, rope consumption was 1 foot per 22.4 tons of ore. Using drums larger than 9 inches, the tonnage per foot of rope was 27.2.

Under average conditions 1 foot of wire rope usually will be worn out per 5 to 15 tons of ore scraped, if sizes of rope, scraper, hoist drums, sheave wheels, and hoist pull are suitably proportioned.

At El Potosi[17] the cost of scraping 107,420 tons of ore in 1932, exclusive of operating labor, was as follows:

Item	*Cost per short ton, in U. S. currency*
Making and repairing scrapers	$0. 0054
Power lines	. 0004
Hoist installations	. 0056
Hoist repairs	. 0167
New cable and cable repairs	. 0038
Electric power	. 0056
Lubrication	. 0014
	0. 0389

Hoist repairs probably would have been less had larger hoists been used. Those actually employed were of 6.5-, 7.5-, 10-, and 15-horsepower capacity.

At Nkana[18] scraping costs in open stopes for a 6-month period in 1935 were as follows:

	Cost per ton	
	Shilling	*Dollar*
European salaries and wages	0. 17	0. 04
Native labor	. 08	. 02
Supplies, including explosives	. 22	. 05
Workshops	. 03	. 01
Underground mechanics and electricians	. 07	. 02
Electric power	. 03	. 01
	. 60	. 15

In recent years there has been a notable increase in the use of scrapers for transferring broken ore from a series of stope raises into an ore pass leading to the haulage level. This practice often greatly reduces the number of long raises that otherwise would have to be driven in the wall rock and eliminates loading and tramming on sublevels. The stope raises are driven from one or both sides of the transfer drifts rather than directly above, so that as the ore piles up it will not block the drift in assuming its natural angle of repose but will leave room over the tops of the sloping piles for passage of the scraper. The raises may be timbered or untimbered but are left open at the bottom. If the ore is sticky and tends to pack, this system is particularly advantageous, since the ore can run freely through the raise and out into the transfer drift, thus eliminating troubles caused by "hang-ups," which are common in chutes equipped with gates.

As indicated by the figures in table 30, large scrapers often can handle 50 to 75 tons per hour or more in this kind of work, only one or two men being required for the operation; 100 to 200 tons per hour have been scraped through transfer drifts under favorable conditions. Scraping distances are usually under 150 to 200 feet but doubtless with suitably designed equipment considerably greater distances are economical.

CONVEYOR SYSTEMS

At present (October 1938) very few data on underground conveyor systems in metal mines are available.

At the La Rue mine on the Mesabi range, Minnesota, belts were installed some time ago on an old haulage level 15 feet below the bottom of the ore. Scrapers deliver the ore from the top-slice stopes to chutes, from which it drops through gates onto the 30-inch belt. The conveyor system is in seven sections and delivers the ore from the 125-foot level to the washing plant on the surface. Although no data are available, it was stated that costs were considerably below those for tramming and hoisting.

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[17] Walker, Harlan A., Mining Methods and Costs at El Potosi mine, El Potosi Mining Co., Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6804, 1934, 38 pp.

[18] Harrison, A. Royden, and Mackay, Kenneth E., Mining Practice at Nkana. II: Eng. and Min. Jour., vol. 137, No. 1, January 1936, p. 17.

At the Spruce mine on the Mesabi range, a conveyor system was installed during the summer of 1937 to handle open-pit ore. Part of the ore is mined by electric shovels that load into trucks, which deliver it to a raise driven from an underground level to the bottom of the pit. The rest is mined by two tower excavators using 3-yard drag buckets, which will operate within a maximum clean-up radius of 700 feet, the average digging radius being 170 feet. Each tower surmounts a raise to which the scraper delivers the ore.

At the top of each of the three raises the ore passes through a crusher before it enters the raise proper and thence falls to a pan-feeder, which feeds the belt. The belt system, fed from the three raises, is 4,991 feet long and in nine sections, each driven by a 75-horsepower, 440-volt, A. C. motor. The belt is 7-ply and 30 inches wide. Belt speed is 500 feet per minute, at which rate the system has a rated capacity of over 500 tons per hour. (It is reported that actual capacity exceeds this considerably.) The maximum adverse belt slope is 30 percent, or 17°. By means of this sytem ore is delivered directly to shipping bins on surface about 1,000 feet from the top of the incline that connects with the mine level. Although this system is part of an open-pit operation, about 70 percent of the belt is underground, and it is mentioned as an example of the suitability and capacity of conveyor belts and to call attention to their probable future wide application to underground transportation, where large tonnages of ore must be handled.

Another underground conveyor belt that was also used in connection with open-pit mining operations was installed at the New Idria mine.[19] This is probably one of the earliest installations in an underground metal mine. It was employed principally to convey waste from the hanging wall of the ore deposit to a waste dump. The waste is blasted or caved into a raise leading to an underground grizzly chamber 27 feet above the 300-foot adit level. From the grizzly it drops onto an apron feeder 42 inches wide by 11 feet long, which feeds it onto the conveyor belt at the rate of about 350 tons per hour, although 1,000 tons per hour can be handled without difficulty. To prevent large boulders from cutting the belt, a steel plate with round holes 1 inch in diameter is placed at the end of the feeder at an angle of 38°. The fine material falls through the plate onto the belt in advance of the large chunks and forms a cushion for the latter to fall upon. The conveyor belt is 42 inches wide, 8-ply, rubber-surfaced 3/16-inch on the top side and 1/32-inch on the bottom. It is 1,250 feet long, center to center of head and tail pulleys; 670 feet is in the adit and the rest extends over the dump on surface. Speed of the belt is 310 feet per minute when waste is being handled. It is driven through a tandem drive, situated midway between the ends, by a 40-horsepower, 850-r. p. m., 440-volt, A. C. motor with spur-gear reduction unit; variable-speed drive is provided so that the belt may be run at 310 feet per minute when waste is handled and 45 feet per minute when ore is handled. At the slow speed, hand-sorting is practiced, the ore being low in grade. At the end of 1930 the belt had been in service 3 years and appeared good for several years more; maintenance expense had been negligible.

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[19] Moorehead, W. R., Methods and Costs of Mining Quicksilver at the New Idria Mine, San Benito County, Calif.: Bureau of Mines Inf. Circ. 6462, 1931, 14 pp.

LOADING CHUTES

Gravity usually is employed wherever possible to move broken ore from stopes to the haulage levels, and ore-passes from the stopes terminate at the bottom in chutes, by means of which the flow of the ore into cars or onto conveyors can be controlled. An exception is where scraper haulage is employed on the level or on transfer levels, in which instance the raises are open at the bottom and the falling ore simply piles up in the scraping drift, as previously described.

Loading chutes are constructed in various ways and consist essentially of a framework of timber, steel, or concrete with gates or other means provided for holding back the column of broken ore in the raise above.

Chute gates are of several different types and have been discussed by Jackson and Knaebel in an earlier Bureau of Mines publication.[20]

Loading of cars from chutes is the first operation in underground haulage, and it is therefore appropriate to discuss chutes under the heading of "Transportation."

Following is a classification of underground chutes and gates that includes the principal types in common use.

1. Platform chutes.
   - *A*. Chinaman chutes.
   - *B*. Scraper loading chutes.
2. Simple stopboard gates.
   - *A*. Cleated; boards between sides of chute.
   - *B*. Projecting; boards supported by iron brackets.
3. Vertical overcut or "guillotine."
   - *A*. Operated by hand lever.
   - *B*. Air-operated.
   - *C*. Rack-and-pinion type.
4. Sliding undercut gates.
5. Hinged gates.
   - *A*. Hinged-lip apron.
   - *B*. Bottom-discharge door; toggle or levers and counterweight.
   - *C*. Front-discharge door; toggle or levers and counterweight.
6. Butterfly gate.
7. Arc-type gates.
   - *A*. Overcut gates.
     - *a*. Flat door.
     - *b*. Curved door; (1) convex face against ore, (2) concave face against ore.
   - *B*. Undercut gates.
     - *a*. Convex side against ore.
     - *b*. Concave side against ore.
8. Finger chutes.
9. Curtain type (flexible door).

A common form of platform or "Chinaman" chute is illustrated in figure 69 at *A*. Strictly speaking, it is not a chute at all, the ore is held back by an offset, as shown, a car is run underneath, and the ore is shoveled or scraped into it through an opening made by removing boards or lagging over the drift timbers. It has the advantage of simplicity, low first cost, and ease of installation. Large chunks can be rolled to one side and broken, and there is no obstruction at the

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[20] Jackson, Chas. F., and Knaebel, John B., Underground Chute Gates in Metal Mines: Bureau of Mines Inf. Circ. 6495, 1931, 22 pp.

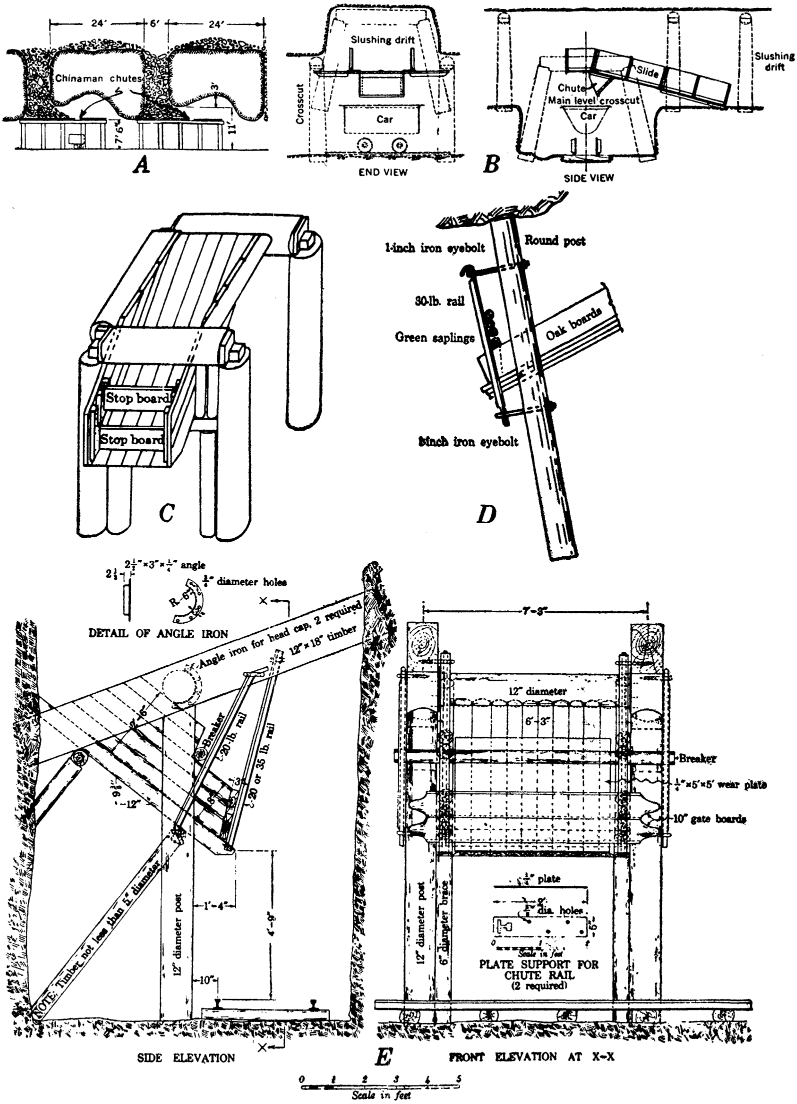
24′
6′
24′
Chinaman chutes
3′
7′6″
11′
A
Slushing drift
Crosscut
Car
END VIEW
B
Slushing drift
Slide
Chute
Main level crosscut
Car
SIDE VIEW
Stop board
Stop board
C
1-inch iron eyebolt
Round post
30-lb. rail
Green saplings
Oak boards
1-inch iron eyebolt
D
2½″×3″×¼″ angle
2⅛
¾″ diameter holes
R-6″
DETAIL OF ANGLE IRON
X
Angle iron for head cap, 2 required
12″×18″ timber
6″
Breaker
20-lb. rail
20 or 35 lb. rail
3″
9⅜″
12″
1′-4″
4′-9″
12″ diameter post
10″
NOTE: Timber not less than 5″ diameter
SIDE ELEVATION
E
7′-8″
12″ diameter
6′-3″
Breaker
¼″×5′×5′ wear plate
10″ gate boards
⅜″ plate
¾″ dia. holes
5″
Scale in feet
PLATE SUPPORT FOR CHUTE RAIL
(2 required)
12″ diameter post
6″ diameter brace
FRONT ELEVATION AT X-X
0 1 2 3 4 5
Scale in feet

FIGURE 69.—Types of platform and stop-board loading chutes. *A*, Common form of platform or "Chinaman" chute. *B*, Steel cantilever-type scraper loading slide. *C*, Simple cleated stopboard chute. *D*, Stopper of green saplings supported by steel brackets. *E*, Large stopboard chute with rail brackets and stoplogs or "breakers."

bottom of the raise behind which they can wedge and cause a "hang-up" of ore. In addition, an opportunity is afforded for sorting waste on the platform. Figure 69, *B*, shows a steel cantilever loading slide with a short chute for loading cars with a scraper.

Stopboard chutes probably are used more widely than any other type, particularly for handling small or moderate tonnages of ore and waste. They are simple in construction and cheap to install. Stopboard chutes are shown in figure 69 at *C*, *D*, and *E*. The ore is drawn by raising one or more of the stopboards. The form shown at *C* is a cleated chute and that at *D* is of the simplest bracket-supported construction. The latter was employed at the Ahmeek mine in Michigan, where the ore was mined in flat-dipping open stopes and the back stands well, requiring little support along haulageways. The form shown at *E* is one developed after long experience at the Hollinger mine in Ontario, where the ore breaks in various sizes from fines to coarse lumps. When stopboards become worn or broken they can be replaced easily and cheaply. Large chunks may cause trouble, and in removing them fines may run out, bury the car, and cover the track. Sometimes a narrow lipboard is set in brackets to hold back fines after the chute is closed, thus assisting in keeping the track clean. When hang-ups are blasted the boards and chute timbers frequently are broken. In general, stopboard chutes are best suited to conditions where the ore breaks fairly small but does not pack, where the total tonnage to be drawn through each chute is not so great that it will be worn out and have to be replaced one or more times, and where rapid loading is not required. In very heavy ground they have one advantage over steel doors, as they can be trimmed or otherwise altered more readily so as to permit their continued operation when damaged by ground pressure and movement.

The vertical overcut or guillotine-type gate is often employed underground, the more elaborate forms on main ore passes, for opening and closing shaft pockets and skip-measuring pockets, and in connection with other more or less permanent or long-life installations. They are especially adapted to handling fine or medium-size material. Lumpy ore may block them and allow fines to run out underneath unless a lipboard or apron also is provided. Warping of the door or springing of the frame may cause them to bind, and therefore they must be installed in a substantial framework so that distortion will be prevented. When the nature of the material is suitable for handling with this type of gate, rapid loading is possible. Figure 70 shows small, hand-operated, guillotine gates at *A* and *B* as installed to handle comparatively small tonnages. The lower one at *A* was used for loading a skip in an inclined shaft and is provided with a short hinged apron to throw the ore farther out from the lip. When folded back, this apron serves to retain fines after the gate is closed. The upper gate in the same figure is operated by a rack and pinion. The gate at *B* is operated by a lever arm, which is simply used to pry up the gate. For rapid handling of larger tonnages, gates of this type usually are operated by compressed-air cylinders. The cylinder is mounted above the chute and raises and lowers the gate by means of a piston attached to a bridle on the top of the gate. It is apparent, however, that considerably more headroom is required than is available in most mine drifts, and the air-operated guillotine gate is more often employed on skip pockets and large main ore passes than on loading chutes in the drifts.

Sliding undercut gates (fig. 70, *C*), operated by compressed air, are not employed extensively but overcome two of the objections to guillotine gates. First, the gate cuts upward through the stream of ore when it closes, effectually stopping the flow of fines; and since the top of the chute is open, large chunks are merely raised and tumbled one way or the other (back into the chute or into the car). Second, no additional headroom is required for installing the air cylinder. On the other hand, fines can hardly be prevented from working in between the gate and the guides and between the gate and the bottom of the chute, causing the gate to bind.

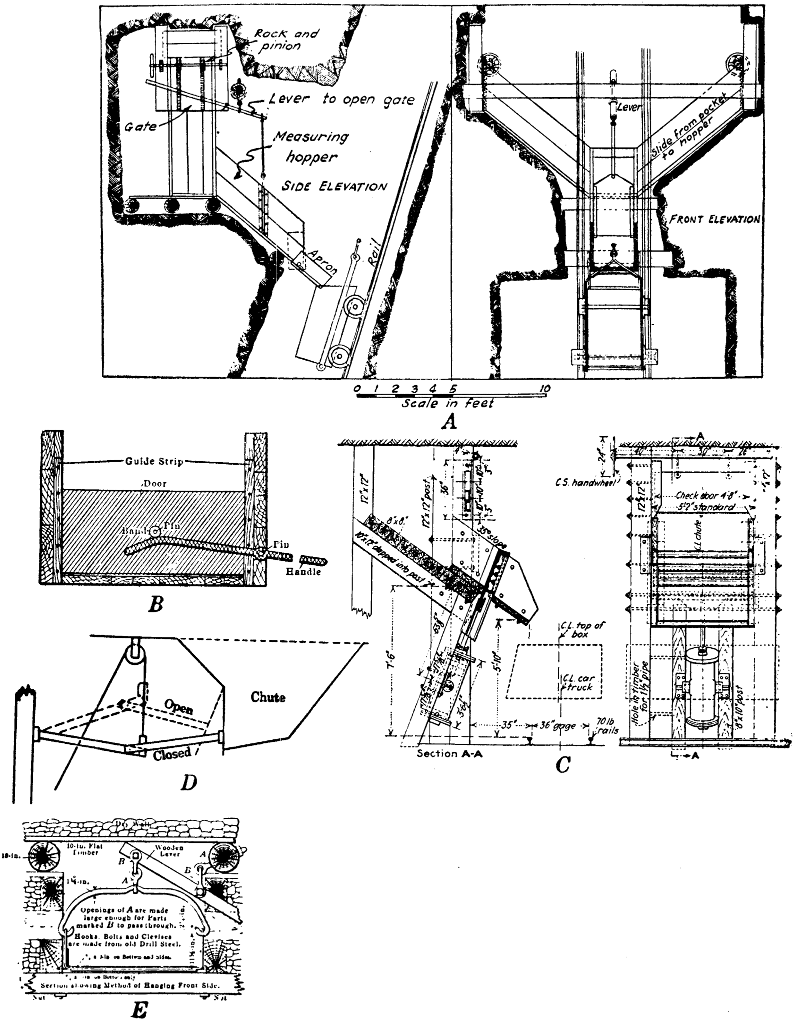
Rock and pinion
Lever to open gate
Gate
Measuring hopper
SIDE ELEVATION
Apron
Rail
Lever
Slide from pocket to hopper
FRONT ELEVATION
0 1 2 3 4 5 10
Scale in feet
A
Guide Strip
Door
Handle
Pin
Pin
Handle
B
12"x12"
12"x12" post
36"
8"x8"
10"x12" dapped into post
35° slope
7'-6"
5'-10"
C.L. top of box
C.L. car truck
35"
36" gage
70 lb rails
Section A-A
C.S. handwheel
24"
40"
30"
26"
Check door 4'8"
5'2" standard
C.L. chute
Hole in timber for 1½" pipe
8"x10" post
A
A
C
Open
Closed
Chute
D
Dry Wall
10-in. Flat Timber
Wooden Lever
18-in.
A
B
B
A
Openings of A are made large enough for Parts marked B to pass through.
Hooks, Bolts and Clevises are made from old Drill Steel.
Section showing Method of Hanging Front Side.
Nut
Nut
E

FIGURE 70.—Types of loading chutes. *A* and *B*, Hand-operated guillotine gates. *C*, Sliding undercut gate. *D*, Front-discharge toggle gate. *E*, Baltic hinged-lip apron.

Hinged gates or doors sometimes are used as lip aprons to catch fines or to throw the material farther out, as in figure 70, *A*. Figure 70, *D*, shows a front-discharge toggle gate hinged at the top.[21] Figure 70, *E*, shows the Baltic hinged gate, which, when dropped, rests on top of the car and serves as a continuation of the chute bottom. When raised, it folds back against the broken ore column. It is operated by a sweep or lever, as shown.

Figure 71, *A*, shows a butterfly-type gate that originated in South Africa. To open it, the stop pin is withdrawn, and the pressure of the ore causes the door plate to swing outward at the bottom; the flow of the ore keeps it open until the loader, having reinserted the stop pin, strikes the under side of the projecting open gate depressing the inside half of the door until the flow of ore catches it and swings it shut against the pin.

Arc gates are of two principal different types—overcut and undercut. Overcut arc gates work best with fine or medium-size dry material. With coarse ore, chunks may lodge under the gate and permit fines to pass because of its failure to close tightly. They are employed extensively, however, and are operated by hand, as at *B* (fig. 71), or by compressed air, as at *C*. The one at *B* is not installed as it should be for greatest ease in opening and closing, however. The overcut gate can be operated more easily by hand if the bottom edge of the arc, when in contact with the bottom of the chute, is slightly above the axis of rotation.

Undercut arc gates are especially adapted to handling large quantities of coarse, hard material. They can be opened and closed quickly, and closing is not hindered by the lodging of chunks under the edge. Figure 72, *A*, shows an undercut arc counterweighted to reduce the effort required for hand operation. Two loaders and one locomotive engineer regularly loaded, trammed, and dumped 300 tons of coarse rock from five chutes of this type in 2½ hours. With this type of gate stop, logs are essential to hold back the column of ore. Figure 72, *B*, illustrates a more elaborate gate operated by compressed-air cylinders. One cylinder operates the undercut gate, and two larger cylinders, one on each side of the chute, operate a check gate (dotted lines). The check gate is of heavy steel construction and is hung on a shaft set between the side posts of the chute. This arrangement was adopted at the Alaska-Juneau mine, where large tonnages were handled through each chute, the life of each installation was long, and the expense was therefore justified. Other forms of undercut gate are employed; in general, those of the arc type are suitable for rapid drawing of large tonnages of chunky ore.

Finger gates (fig. 73) consist of a number of arms or fingers separately suspended on a horizontal shaft above the chute opening, each finger being free to move independently of the others. The simplest form is shown at *A* and is operated by a hand line or by a hand winch. The fingers are pivoted at the knuckle, and the weight of the long (horizontal) arm closes the short arm against the ore. A pull on the rope serves to open all the fingers, but if a lump lodges behind one or more of them they may be pried up separately with a bar, releasing the lump without disturbing the others. A board may be inserted behind the lip angle irons to hold back fine material. A finger gate actuated

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[21] Engineering and Mining Journal, Toggle-Joint Gate: Vol. 88, No. 11, 1909, p. 526.

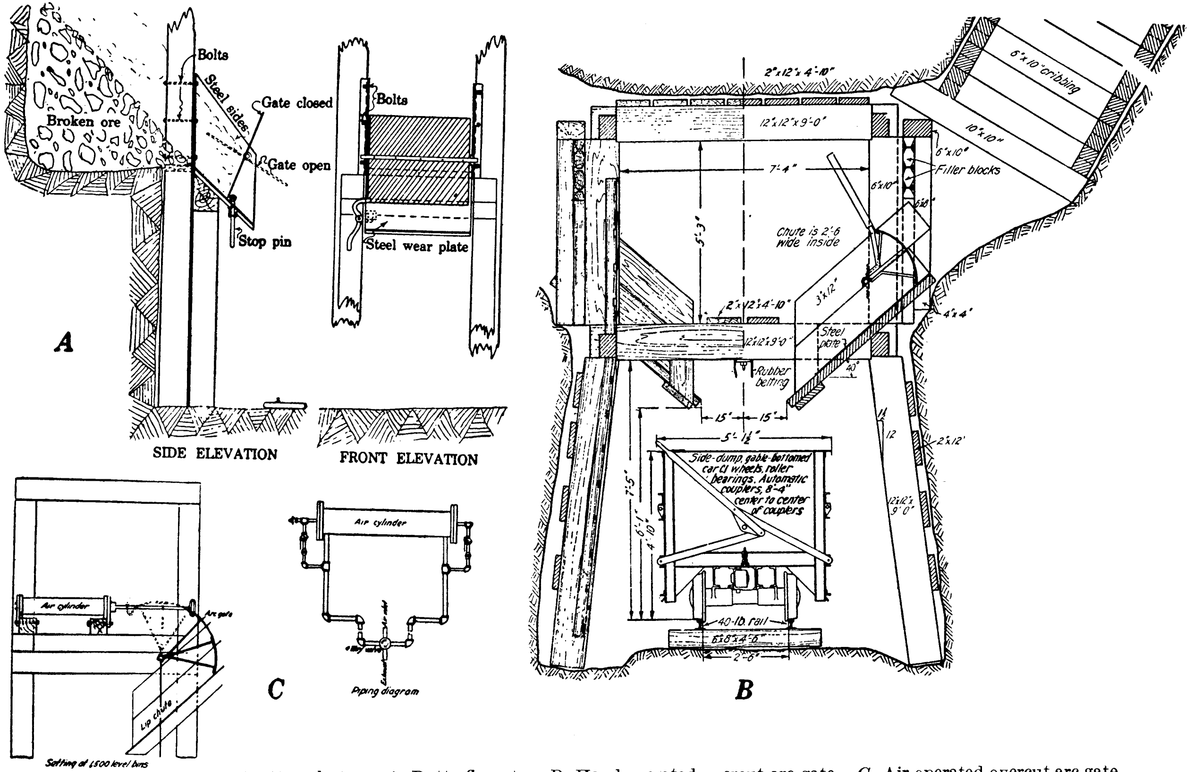
A
Broken ore
Bolts
Steel sides
Gate closed
Gate open
Stop pin
Bolts
Steel wear plate
SIDE ELEVATION
FRONT ELEVATION
B
2"x12"x 4'-10"
6"x10" cribbing
10"x10"
12"x12"x9'-0"
7'-4"
6"x10"
6"x10"
Filler blocks
6"x8"
5'-3"
Chute is 2'-6 wide inside
3"x12"
2"x12"x4'-10"
4"x4"
12"x12"x9'-0"
Steel plate
40°
Rubber belting
15"
15"
1½
12
2"x12'
5'-1½"
Side-dump, gable-bottomed car CI wheels, roller bearings. Automatic couplers, 8'-4" center to center of couplers
7'-5"
6'-1"
4'-10"
12"x12"x 9'-0"
40-lb. rail
6"x8"x4'-6"
2'-6"
C
Air cylinder
Arc gate
Lip chute
Setting at 1500 level bins
Air cylinder
Air inlet
Exhaust
Piping diagram

FIGURE 71.—Types of loading chutes. *A*, Butterfly gate. *B*, Hand-operated overcut arc gate. *C*, Air-operated overcut arc gate.

by an air jack is shown at *B* (fig. 73). It may be noted that the gate shown at *A* requires considerable head room and that at *B* requires room on the opposite side of the drift.

Finger chutes are suitable for handling coarse, blocky ore, and their chief advantage lies in the means afforded for removing large chunks, one at a time (by lifting one or more fingers separately), without causing a run of muck. They are relatively high in cost, cumbersome, and require considerable room for installation.

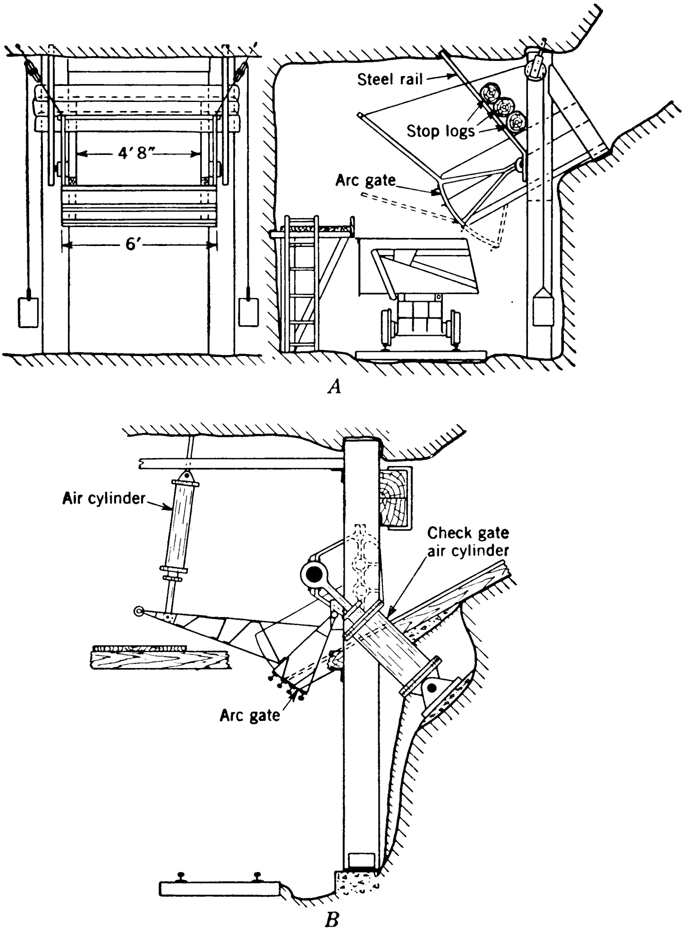

FIGURE 72.—Undercut arc gates. *A*, Hand-operated gate. *B*, Air-operated gate.

The curtain or flexible-door gate is illustrated diagrammatically in figure 74. This is known as the Cramp chain gate and is named for the inventor, D. L. Cramp, mechanical superintendent of the Lake Shore mine, Kirkland Lake, Ontario. It is understood to be protected by patents. It has proved its adaptability to loading either fine or chunky ore, or both together, requires no extra room for its installation, has a long life, and does not require frequent repairs. The balls and chains can be taken off readily and reinstalled quickly on another chute. They are not injured by chunks that may fall against them, because of their flexibility, and for the same reason chunks can be blasted in the chute without damage. There is ample space between the chains for inserting a bar. To open, the balls, which are attached to a yoke, may be raised by hand rope, hand winch, or air piston. Upon closing, the balls, by reason of their shape, fall into position alongside one another and prevent fine material from passing. The balls may be pried up, one at a time, for releasing chunks, as with the finger chutes. A rather flat chute bottom is desirable, although angles of 45° have been satisfactory with some types of rock.

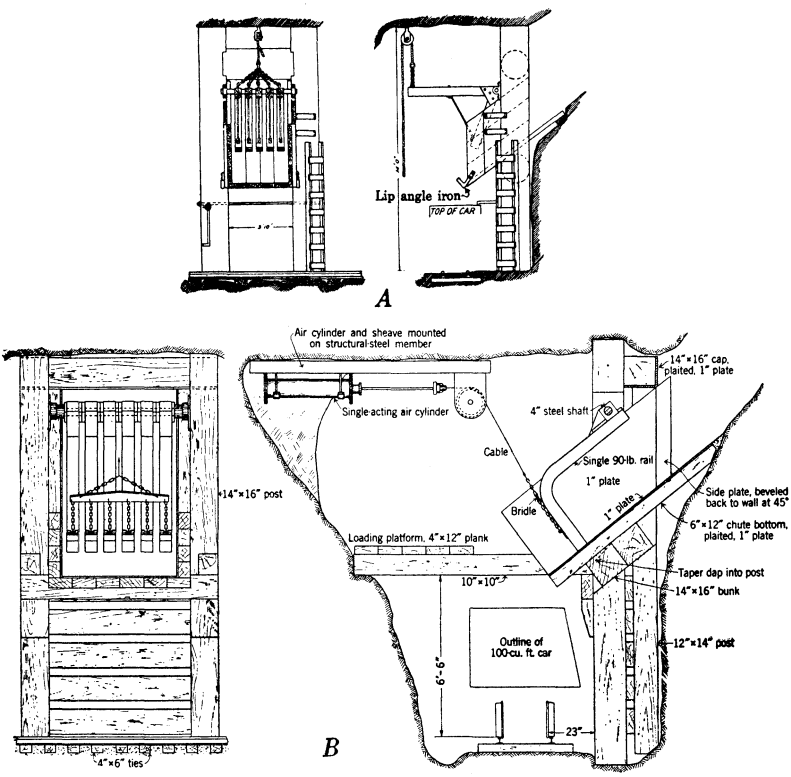
Lip angle iron
TOP OF CAR
A
Air cylinder and sheave mounted on structural-steel member
Single-acting air cylinder
4" steel shaft
14"×16" cap, plaited, 1" plate
Cable
Single 90-lb. rail
1" plate
Bridle
1" plate
Side plate, beveled back to wall at 45°
6"×12" chute bottom, plaited, 1" plate
14"×16" post
Loading platform, 4"×12" plank
10"×10"
Taper dap into post
14"×16" bunk
12"×14" post
Outline of 100-cu. ft. car
6'-6"
23"
4"×6" ties
B

FIGURE 73.—Finger chutes. *A*, Standard finger chute at Engels mine, California. *B*, Air-operated finger chute, Britannia mine, British Columbia.

COST OF INSTALLATION

Table 31 gives typical costs of building and installing loading chutes. The costs shown cover single chutes and are to be considered for the most part as approximate and not directly comparable, since they were obtained from various sources without reference to any standard form for segregation of cost items. Thus, in some cases cost of cutting out ground for timbers was included, whereas in others this item may have been omitted.

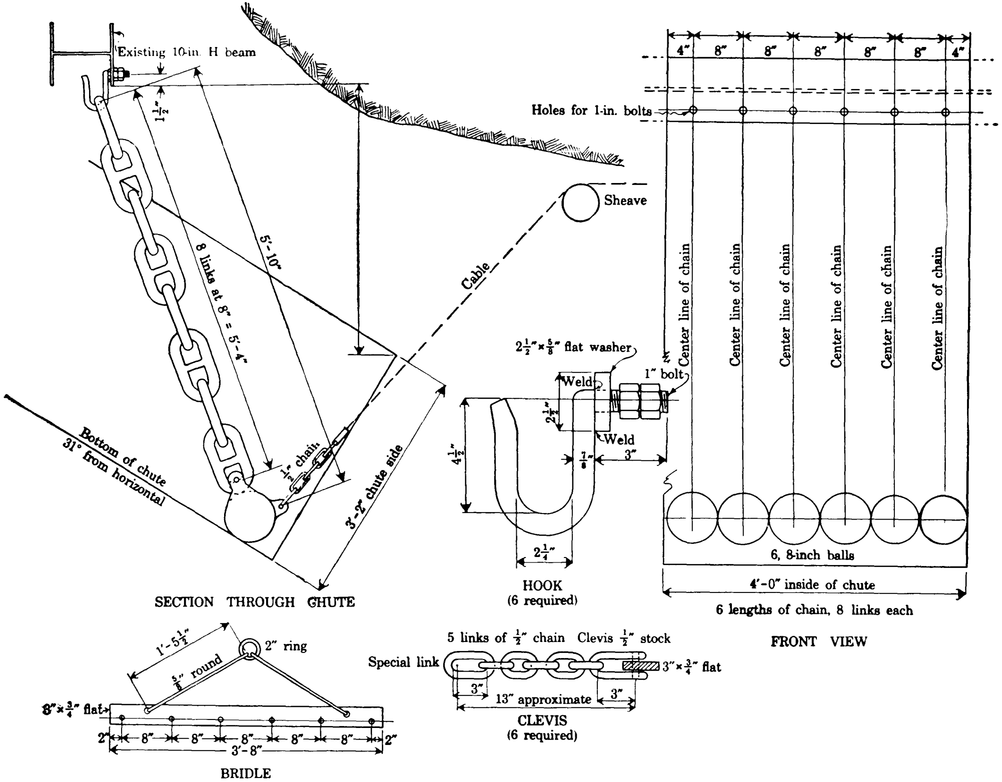
Existing 10-in. H beam
$1\frac{1}{2}''$
8 links at 8" = 5'-4"
5'-10"
Cable
Sheave
$\frac{1}{2}''$ chain
Bottom of chute 31° from horizontal
3'-2" chute side
SECTION THROUGH CHUTE
$2\frac{1}{2}''\times\frac{5}{8}''$ flat washer
Weld
1" bolt
$2\frac{1}{2}''$
$4\frac{1}{2}''$
Weld
$\frac{7}{8}''$
3"
$2\frac{1}{4}''$
HOOK
(6 required)
4" 8" 8" 8" 8" 8" 4"
Holes for 1-in. bolts
Center line of chain
Center line of chain
Center line of chain
Center line of chain
Center line of chain
Center line of chain
6, 8-inch balls
4'-0" inside of chute
6 lengths of chain, 8 links each
FRONT VIEW
$1'-5\frac{1}{2}''$
2" ring
$\frac{5}{8}''$ round
$8''\times\frac{3}{4}''$ flat
2" 8" 8" 8" 8" 8" 2"
3'-8"
BRIDLE
5 links of $\frac{1}{2}''$ chain
Clevis $\frac{1}{2}''$ stock
Special link
$3''\times\frac{3}{4}''$ flat
3"
13" approximate
3"
CLEVIS
(6 required)

FIGURE 74.—Chain gate for 1,800-level ore pass, Lake Shore mine, Ontario.

TABLE 31.—*Typical costs of building and installing loading chutes*

Mine	Type of chute or gate	Cost per chute				
		Labor			Materials	Total
		Making	Installing	Total		
Ahmeek, Mich.	Stop board (fig. 69, *D*)			$5.00	$11.00	$16.00
Hollinger, Ontario	Stop board (fig. 69, *E*)			17.25	35.25	52.50
Negaunee, Mich.	Cleated stop board			13.30	28.80	42.10
Do	Overcut arc	$14.15			49.40	63.55
Pioneer, British Columbia.	Stop board 3 ft. 2 in. wide		$9.90	9.90	6.00	15.90
Baltic, Mich.	Hinged-lip apron (fig. 70, *E*)		7.54		[1] 16.25	23.79
United Verde, Ariz.	Overcut arc [2] 33½ in. wide		35.00		42.00	[3] 77.00
United Verde Extension, Ariz.	do	3.24	4.70	7.94	8.00	15.94
Lake Shore, Ontario	do	6.25	10.00	16.25	15.00	31.25
Inspiration, Ariz.	Overcut arc curved down (fig. 71, *B*) 30 in. wide.		8.50		24.76	[4] 33.26
Cananea, Mexico	Overcut arc curved down (fig. 71, *B*) 42 in. wide.		6.25		[1] 38.69	[5] 44.94
Park-Utah, Utah	Overcut air-operated arc			85.00	30.00	115.00
Engels, Calif.	Finger chute (fig. 73, *A*) 36 in. wide.	16.46	70.00	86.46	57.33	[6] 143.79
Lake Shore, Ontario	Chain gate (fig. 74)	32.00	13.00	45.00	127.00	[7] 172.00

[1] Includes labor of making.
[2] Flat instead of curved plate, but swings through an arc.
[3] Mining and mucking labor, $50; explosives, $20 additional.
[4] Salvage value for re-use, $13.14.
[5] Total cost, including cost of raise to top of pony set, 4 feet above rail is $192.23.
[6] Cost of excavating chute raise and station, $131.63.
[7] Does not include cutting raise or the 10-inch beam cap piece. Salvage value for re-use is high.

ORE AND WASTE PASSES AND TRANSFERS

Ore passes are simply vertical or inclined openings for transfer of broken ore to the haulage system. They may be raises in the ore or in the walls of the deposit or timbered openings through filled stopes and, as previously stated, usually terminate at the bottom in some form of loading chute.

They must be steep enough to insure that the ore will run freely and usually are inclined at angles ranging from 50° to vertical. The inclination often is determined largely by the dip of the ore body, especially if it is a narrow, tabular one, but the physical characteristics of the broken ore also must be considered. Most ores will run well at 60° to 65°. Some hard, dry, lumpy ores will run at angles as low as 45° if the cross-sectional area of the pass is large, but most ores will not. Moist, sticky ores are especially troublesome; the fines often will build up on the bottoms of flat raises and, if the raise is too steep, will pack owing to the superincumbent load, if much ore is allowed to accumulate, that is, unless the raise is kept nearly empty. In vertical or steep ore passes, chunks of hard ore falling considerable distances are likely to break the chute gates unless some ore is left in the bottom at all times. The fall in long ore passes may be broken by short offsets or by reversing the dip. Thus, it is common practice to drive an ore-pass raise at 50° to 60° for 50, 75, or 100 feet and then "knuckle back" on a slope in the opposite direction. This often assists materially in reducing wear on ore-pass linings and preventing breakage of chutes and gates.

The type of lining will depend upon the nature of the surrounding material (whether rock, ore, or waste filling), the characteristics of the material to be handled, and the total quantity of ore to be transferred during the life of the pass. In hard firm ore and rock, ore and waste passes often are unlined.

In soft ore, lined sets or cribbing may be employed to support the surrounding ground and to take the wear. The lining usually consists of planking laid the long way—that is, in the direction of movement of the ore—and is secured to the timber sets or cribbing by spikes. Falling chunks often cause trouble by striking the planking a glancing blow and knocking it off, and under such conditions it may be necessary to employ lag screws instead of spikes. An alternative to plank lining employed successfully under these conditions at some mines is to use heavy sawed timber for cribbing, the timbers being laid "skin-to-skin" to form a smooth inside face. If the life of the ore pass is to be long and a large tonnage is to be handled through it, the cost of concrete lining may be justified. This would seldom be the case for individual stope chutes but may be for main ore transfers that are to be in service for a number of years. Sometimes T-rails instead of planking are laid lengthwise of the ore pass to take up the wear and protect the timbers.

The ore passes should be of cross-sectional area adequate for the tonnage to be handled and the size and character of the ore. If too small, chunks are likely to wedge themselves between the sides and block the pass. Since vexatious and costly delays and broken chutes often result from "hung" ore passes, it is poor economy to restrict their size. It is customary to carry a manway along one side of the ore pass as a separate compartment of a two-compartment raise, and access to points where the ore pass becomes blocked may be had from the manway.

Waste passes for introduction of stope filling often are driven from one level to the next before stoping operations are begun but may be driven from the backs of the stopes after they have been mined up part way. As with ore passes, they may be timbered or not, depending on the character of the ground, but passes to individual stopes are used for a comparatively short time (until the stope is mined out and filled), and permanence of construction usually is not required. As with ore passes, the inclination and size must be adapted to the nature of the filling material.

The passes may terminate in the stope at a chute or may be open at the bottom, depending on the method of distributing the fill in the stopes. If it is to be spread by natural gravity flow or with scrapers, the passes are open; but if it is to be drawn into stope cars for distribution, chute gates must be used. The chutes in this case should be of one of the simpler forms such as the stop-board type, easily and quickly installed, since they must be moved frequently as the stope advances upward.

MAIN TRANSFERS

As previously pointed out, considerable savings can sometimes be made in cost of development if only certain levels are connected with the shaft and intermediate levels are used solely for ore extraction and transfer purposes. Even though all the levels connect with the shaft, savings can often be made by providing for hoisting only from one or more main haulage levels to which the ore from intermediate levels is transferred through ore passes. In this way the cost of installing, operating, and maintaining ore-storage and skip-loading pockets on the intervening levels is eliminated, and additional savings may be possible through concentration of main-line haulage on one or relatively few levels.

This practice requires that one or more main ore passes be provided for transferring the ore from the intermediate or ore-extraction levels to the haulage level below. The position of these passes will obviously be determined by the relative positions of the ore bodies, intermediate levels and shafts, or other main outlets of the mine. Since each transfer of ore costs money, it is desirable to handle it the minimum number of times; hence, wherever possible, ore passes connecting each extraction level directly with the main haulage level should be provided.

A practice that has become more and more common in recent years is to provide a main ore pass paralleling the shaft, into which ore can be dumped on any ore-extraction level and drawn off at any hoisting level below. In addition, where a filled-stope method of mining is employed it is advantageous to have a main waste pass into which waste may be dumped and from which it may be drawn on each level. Even if other than a filling method is employed, a main waste pass paralleling the shaft may be advantageous. If a filling system is used the waste pass or passes may be either close to the shaft on all levels or some distance from the shaft and centrally situated with respect to the stoping areas.

Figure 75 shows ore- and waste-pass systems paralleling the hoisting shafts, by which ore may be dumped, and waste may be either dumped or drawn off on every connecting level.

It should be noted that the fall of ore and waste is broken at each level by an offset in the pass or by "knuckling back" on the dip; this also keeps the passes close to the shaft and at the same time at an angle from the vertical.

Figure 76 shows an ore- and waste-pass system in which there is a main ore pass and one main waste pass approximately parallel to the shaft and other main waste passes (shown in heavy black) are provided in the stoping areas.

Main ore passes often remain in service for years and large tonnages may be handled through them. For this reason considerable expense may be justified for providing permanent resistant linings with chutes and gates designed for optimum operating efficiency. In addition to their other functions, main ore and waste passes serve a useful purpose in providing storage reservoirs. Thus in the event of delays on the haulage level or in the shaft or of break-downs in the milling plant, they may obviate suspension of mining in the stopes by reason of their capacity to store the broken ore until normal operations can be resumed. Likewise, if for any reason stoping must be discontinued for a short time, they may serve to provide ore for the mill during the interim.

Stope-filling material often is required in large quantities at regular or irregular intervals. To provide this when needed and thus avoid delays in the stopes, main waste passes can be utilized for accumulation of waste during periods when the demand for waste is small.

At the Hollinger mine in Ontario [22] a system of ore and waste passes parallels the Central shaft from surface to the skip-loading pockets below the 2,750-foot-level station, and another system parallels the No. 19 shaft from the 2,600-foot level to loading pockets below the 3,800-foot level. Ore can be dumped into the Central shaft system on any level from the 2,750 up, and waste can be dumped into or drawn from the waste pass at any level. The ore-pass system at this shaft together with the storage pockets has a capacity of 25,000 tons, enough for about a 5-day run for the mill. The waste-pass system has a storage capacity of 6,500 tons.

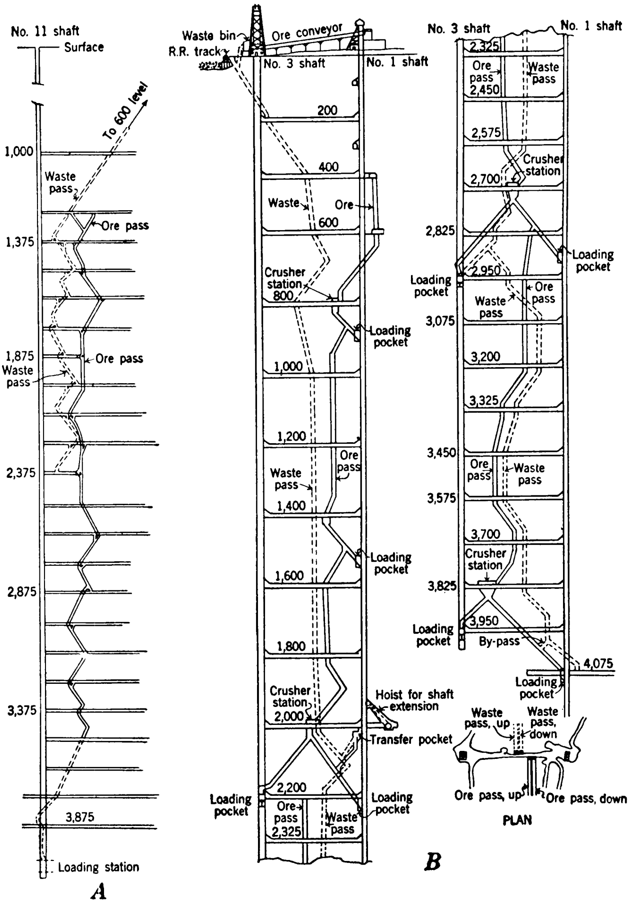
No. 11 shaft
Surface
To 600 level
1,000
Waste pass
Ore pass
1,375
1,875
Waste pass
Ore pass
2,375
2,875
3,375
3,875
Loading station
A
Waste bin
R.R. track
Ore conveyor
No. 3 shaft
No. 1 shaft
200
400
Waste
Ore
600
Crusher station
800
Loading pocket
1,000
1,200
Ore pass
Waste pass
1,400
Loading pocket
1,600
1,800
Crusher station
2,000
Hoist for shaft extension
Transfer pocket
2,200
Loading pocket
Ore pass
Waste pass
2,325
Loading pocket
B
No. 3 shaft
No. 1 shaft
2,325
Ore pass
Waste pass
2,450
2,575
Crusher station
2,700
2,825
Loading pocket
2,950
Loading pocket
Ore pass
Waste pass
3,075
3,200
3,325
3,450
Ore pass
Waste pass
3,575
3,700
Crusher station
3,825
3,950
Loading pocket
By-pass
4,075
Loading pocket
Waste pass, up
Waste pass, down
Ore pass, up
Ore pass, down
PLAN

FIGURE 75.—Ore- and waste-pass systems. *A*, System at McIntyre mine, Ontario. *B*, System at Lake Shore mine, Ontario.

SHAFT STATIONS, STORAGE, AND LOADING POCKETS

Arrangements for handling ore and waste at the shaft will depend on the method of hoisting (whether in buckets, in cars on cages, or in skips); the tonnage and physical characteristics of the ore; whether two or more classes of ore must be hoisted and kept separate; the method employed for transporting the ore on the levels and the type and size of cars; and the inclination of the shaft.

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[22] Young, Arthur W., Mining [Hollinger Consolidated Gold Mines, Ltd.]: Canadian Min. Jour., September 1935, p. 377.

HOISTING IN BUCKETS

In mines having a small output the ore may be hoisted in buckets such as those already discussed under the caption, "Equipment, Methods, and Costs of Shaft Sinking." Even relatively large tonnages may be handled in this way, as in the Tri-State zinc and lead district, where buckets or "cans," as they are termed locally, have been employed for years for hoisting 300 or 400 tons or more per shift; it is reliably reported that 150 cans or 90 tons per hour can be hoisted by this method.

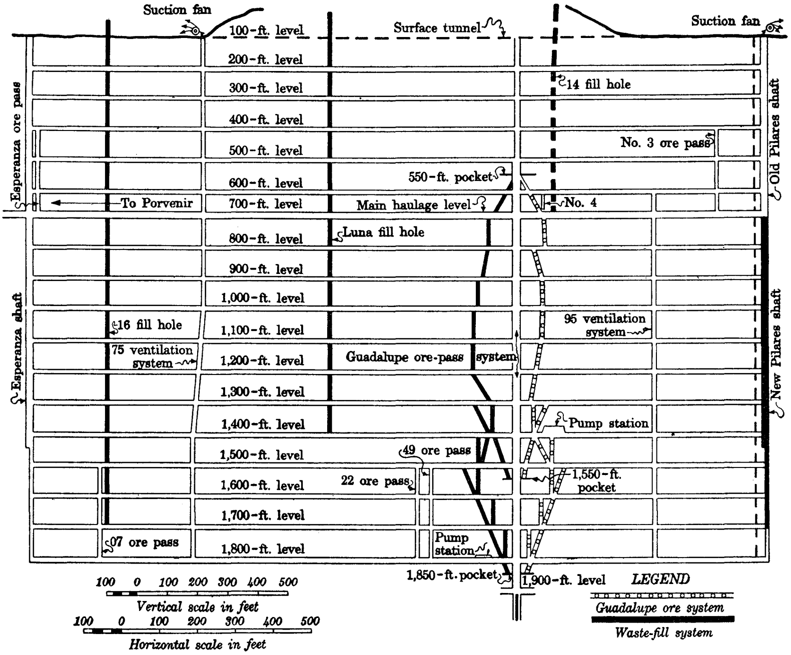

Suction fan
100-ft. level
Surface tunnel
Suction fan
200-ft. level
300-ft. level
14 fill hole
400-ft. level
500-ft. level
No. 3 ore pass
600-ft. level
550-ft. pocket
Esperanza ore pass
Old Pilares shaft
To Porvenir
700-ft. level
Main haulage level
No. 4
800-ft. level
Luna fill hole
900-ft. level
1,000-ft. level
16 fill hole
1,100-ft. level
95 ventilation system
75 ventilation system
1,200-ft. level
Guadalupe ore-pass system
Esperanza shaft
New Pilares shaft
1,300-ft. level
1,400-ft. level
Pump station
1,500-ft. level
49 ore pass
1,600-ft. level
22 ore pass
1,550-ft. pocket
1,700-ft. level
07 ore pass
1,800-ft. level
Pump station
1,850-ft. pocket
1,900-ft. level
100 0 100 200 300 400 500
Vertical scale in feet
100 0 100 200 300 400 500
Horizontal scale in feet
LEGEND
Guadalupe ore system
Waste-fill system

FIGURE 76.—Diagrammatic longitudinal vertical section showing ore- and waste-pass systems, Pilares mine Mexico.

In this district the practice has been to use cans of 1,200 to 1,400 pounds capacity, which are transported to the faces of the flat stopes on small four-wheel, flat-top trucks running on light steel rails. When loaded, these cans are trammed, usually by hand or mule, to a siding, where they are made up into trains and hauled to the shaft bottom (fig. 77, *A*). Here men known as bumpers uncouple the trucks and push them, one at a time, to a spot directly under the center of the shaft.

A man called a hooker is stationed at this point; it is his duty to detach the empty down-coming buckets and hook the loaded buckets on the hoisting rope. A hook is attached to the end of the wire hoisting cable, and as the empty bucket or can comes within reach

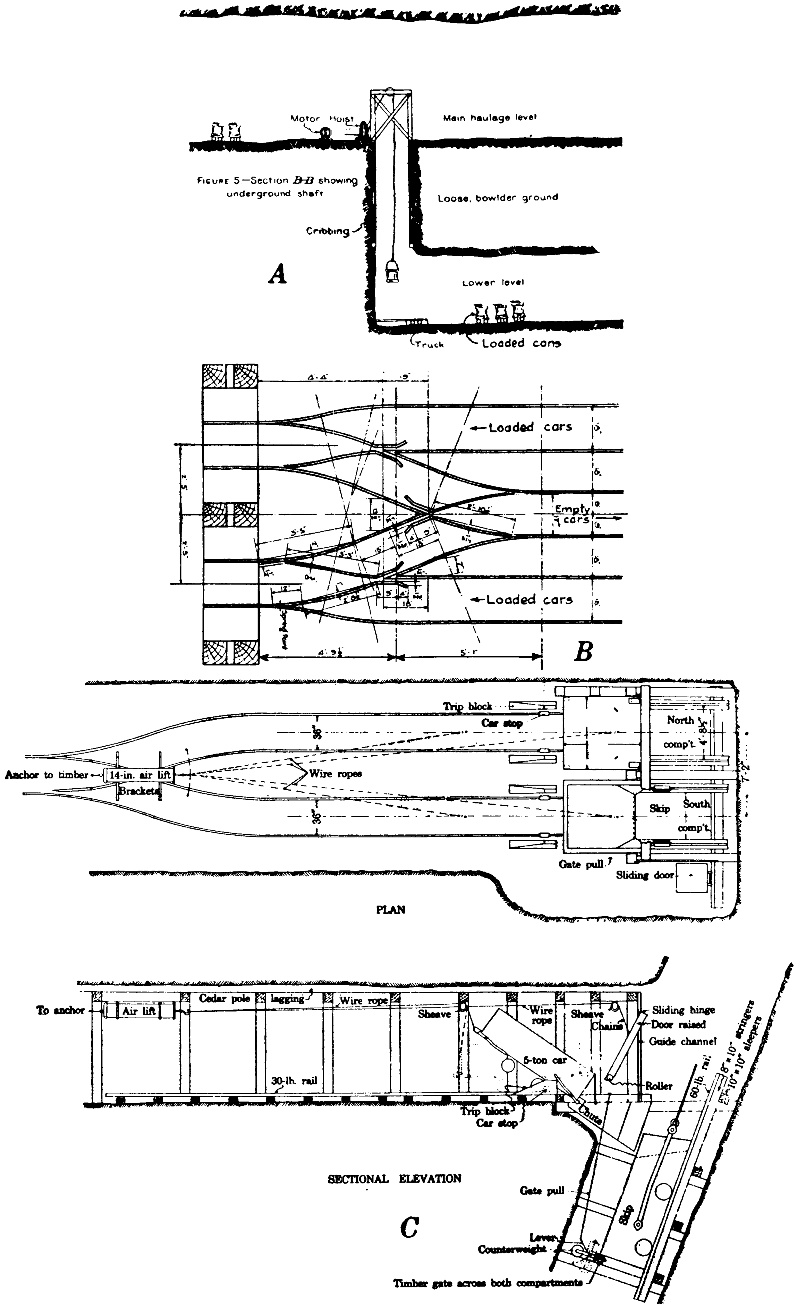

Motor Hoist
Main haulage level
Figure 5.—Section B-B showing underground shaft
Loose, bowlder ground
Cribbing
A
Lower level
Truck
Loaded cans
Loaded cars
Empty cars
Loaded cars
B
Trip block
Car stop
North comp't.
Anchor to timber
14-in. air lift
Brackets
Wire ropes
Skip
South comp't.
Gate pull
Sliding door
PLAN
To anchor
Air lift
Cedar pole
lagging
Wire rope
Sheave
Wire rope
Sheave
Chains
Sliding hinge
Door raised
Guide channel
6-ton car
30-lb. rail
Roller
60-lb. rail
8" x 10" stringers
10" x 10" sleepers
Trip block
Car stop
Chute
SECTIONAL ELEVATION
Gate pull
Skip
C
Lever
Counterweight
Timber gate across both compartments

FIGURE 77.—*A*, Shaft bottom for hoisting ore in cans, Tri-State district. *B*, Station-track lay-out for caging cars on one side of shaft. *C*, Station lay-out for dumping directly into skip, cars dumped by rope and air lift.

of the hooker he swings it to one side and lowers it onto an empty truck placed there by the bumper. He quickly detaches the hook from the empty and attaches it to the loaded can; it is then hoisted to surface, where it is dumped by the hoist operator and returned to the shaft bottom. The hoistman and hooker work as a team and become so proficient that no signals are required as long as there is a supply of loaded cans at the bottom. The hoist is mounted in the head frame over the shaft, and the hoistman can tell by the behavior of the rope just what is going on at the bottom. The hooker becomes so adept that the unhooking of the empty and hooking of the loaded can are accomplished in one motion.

In some small mines the ore is trammed to the shaft in small cars and either dumped directly into the bucket or into a small loading pocket provided with a chute and gate, whence it is drawn into the bucket. When dumped direct, a hinged apron or spout is required to conduct the ore into the bucket without spilling.

HOISTING IN CARS ON CAGES

Although hoisting in skips has largely replaced hoisting in cars at most modern metal mines, the latter practice is still employed and under some conditions has certain advantages that may outweigh the disadvantages. Thus in mines where two or more grades or classes of ore in addition to waste must be hoisted separately, it would be necessary, when using skips, either to have a separate pocket for each grade of ore and another for waste or, if there is only one pocket, to clean it carefully when changing from one grade or class to another. The disadvantages of the latter practice are more apparent when rapid hoisting is required, and it may be necessary to provide long sidings at the stations on which to assemble cars according to classes of ore. This calls for additional excavation to make room for necessary trackage and involves investment in more cars than would be needed otherwise. If separate pockets are provided, and especially if they have to be duplicated on each of several hoisting levels, the cost of preparation is high, and if ground conditions around the shaft are bad the excavations may weaken the shaft considerably.

Timber, drills, steel, explosives, and other mine supplies and equipment usually can be taken underground more easily in cars or on flat trucks than in a skip. Another advantage sometimes claimed for hoisting ore and waste in cars on cages is that if the shaft is not too busy the same cages may be used for hoisting ore as for handling men and supplies.

Arrangements for caging cars at the shaft stations vary. In some mines stations are cut on both sides of the shaft. The loaded cars are fed onto the cage on one side and push the empty car off on the other side, an arrangement common in coal mines but less so in metal mines. The loads may run down to the shaft by gravity or be pushed by hand or mechanical means on a level track. When run by gravity it is necessary to provide automatic car stops on the track close to the shaft to prevent movement of the cars except when they are intentionally released by the cager.

In other mines the station is cut only on one side of the shaft so that the empty car must be pulled back off the cage before the loaded car can be pushed on. Where space is at a premium, the floor of the station may be covered with steel plates (turn sheets), on which the cars may be shifted about without using tracks. This practice was common for many years and is still used to some extent, especially in development work. Where more room is available or can be made so, the work of the cagers is lightened by bringing the mine tracks up to the shaft in alinement with the tracks on the cages (fig. 77, *B*). Switches and cross-overs are provided for pulling the empties off to one side to allow caging the loaded cars. Besides lightening the the labor, the track system is safer; turn sheets become wet and slippery and are likely to cause bad falls.

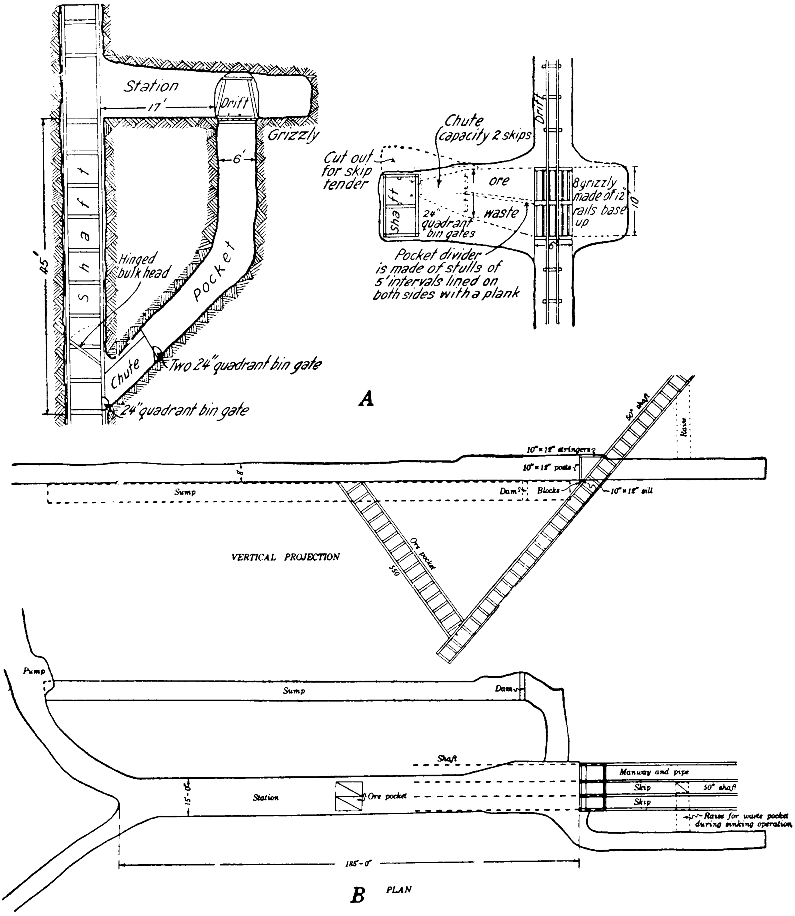
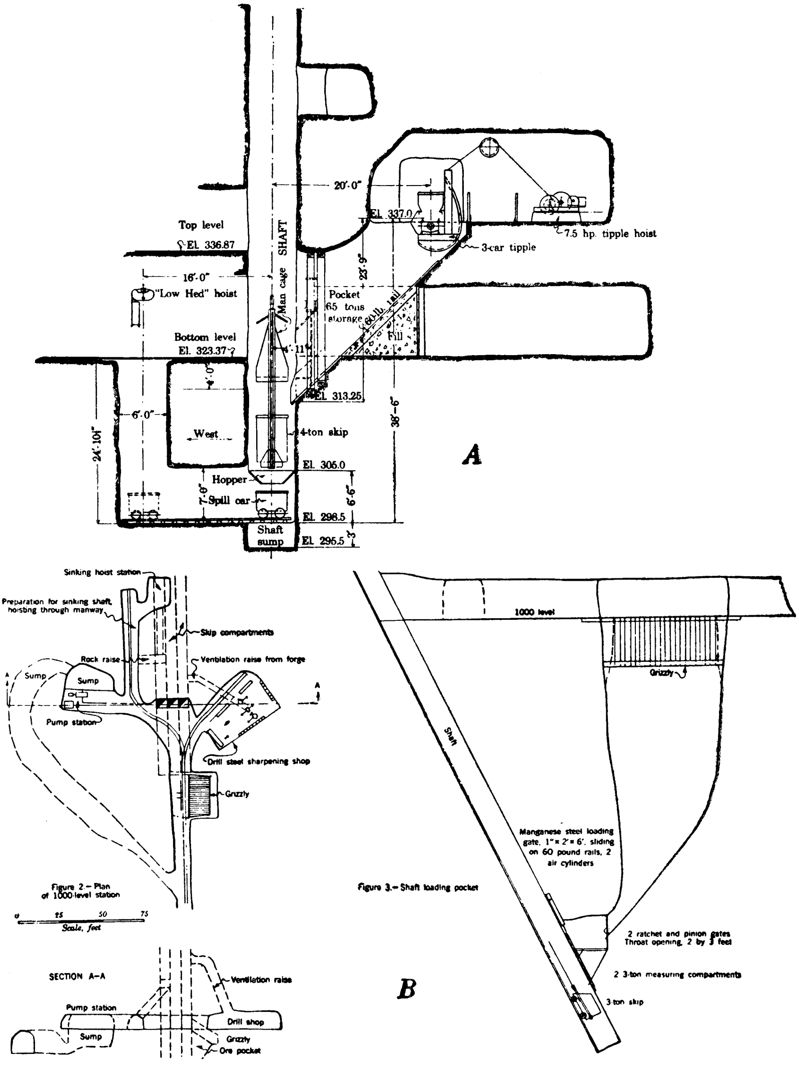

Station
17′
Drift
Grizzly
6′
45′
Shaft
Hinged bulk head
pocket
Chute
Two 24″ quadrant bin gate
24″ quadrant bin gate
Chute capacity 2 skips
Cut out for skip tender
Shaft
ore
waste
24″ quadrant bin gates
8 grizzly made of 12 rails base up
10′
Drift
Pocket divider is made of stulls of 5′ intervals lined on both sides with a plank
A
50° shaft
Raise
10″ × 12″ stringers
10″ × 12″ posts
8′
Sump
Dam
Blocks
10″ × 12″ sill
VERTICAL PROJECTION
Ore pocket
550
Pump
Sump
Dam
Shaft
Manway and pipe
Skip
50° shaft
Skip
Station
15′-0″
Ore pocket
Raise for waste pocket during sinking operation
185′-0″
B
PLAN

FIGURE 78.—Simple types of skip pockets and shaft stations. *A*, For vertical shaft. *B*, For inclined shaft.

Hoisting in cages is considerably slower than hoisting with skips, due to the smaller net load that can be hoisted in a shaft of the same size and to the time required for caging cars at the shaft stations and on surface and usually requires more labor.

HOISTING IN SKIPS

Skip hoisting now is employed commonly in metal mines after they have reached the regular production stage. Storage pockets and loading chutes usually are provided at the shaft, although where the output is small and the shaft is not overloaded, mine cars may be dumped directly into the skips, a short chute or apron being used to direct the flow of the ore; the chute can be hinged to fold up out of the shaft after the skip has been loaded. Figure 77, *C*, shows the plan and elevation of a station where cars are dumped directly into the skip.



20′-0″
El. 337.0
7.5 hp. tipple hoist
3-car tipple
Top level
El. 336.87
SHAFT
Man cage
16′-0″
"Low Hed" hoist
23′-9″
Pocket
65 tons
storage
60-lb. rail
Fill
Bottom level
El. 323.37
4′-11″
4′-0″
El. 313.25
38′-6″
6′-0″
24′-10½″
West
4-ton skip
El. 305.0
Hopper
7′-8″
Spill car
6′-6″
El. 298.5
Shaft sump
El. 295.5
3′
A
Sinking hoist station
Preparation for sinking shaft, hoisting through manway
Skip compartments
Rock raise
Ventilation raise from forge
Sump
Sump
A
A
Pump station
Drill steel sharpening shop
Grizzly
Figure 2.—Plan of 1000-level station
0 25 50 75
Scale, feet
SECTION A–A
Ventilation raise
Pump station
Drill shop
Sump
Grizzly
Ore pocket
1000 level
Grizzly
Shaft
Manganese steel loading gate, 1″ × 2′ × 6′, sliding on 60 pound rails, 2 air cylinders
Figure 3.—Shaft loading pocket
2 ratchet and pinion gates Throat opening, 2 by 3 feet
2 3-ton measuring compartments
3-ton skip
B

FIGURE 79.—*A*, Skip pocket with gate for drawing ore directly into skip. *B*, Pocket for inclined shaft with measuring hopper for loading the skip.

For large output, storage pockets are essential, and skip-loading devices greatly expedite hoisting and speed the entire operation. Pockets should be large enough to permit immediate dumping of cars or trains and their prompt return to the chutes for reloading.

Figure 78, *A*, shows a plan and section of a simple type of skip pocket and shaft station. The ore cars are dumped into the pocket, which is covered by grizzly bars on which large chunks are caught and broken before they enter the pocket. A minimum amount of excavation is required and the shaft is not weakened by this style of pocket, but to obtain storage capacity the loading station must be considerably below the level. In this particular installation ore is hoisted in only one compartment of the shaft. Figure 78, *B*, shows a similar arrangement at an inclined shaft.

Figure 79, *A*, shows a loading pocket with the front of the bin nearly flush with the side of the shaft. The ore is drawn directly from the pocket into the skip without the use of a measuring hopper. The haulageway runs parallel to the long dimension of the shaft, and as the cars, which are of the solid-body type, come over the pocket they are dumped by means of a rotary dump. The ore is drawn through vertical, compressed-air-operated, overcut guillotine gates.

Figure 79, *B*, shows a lay-out for an inclined shaft where the cars are dumped into a large pocket and after dumping are pushed back onto a tail track. The ore is drawn from the pocket through rack-and-pinion, overcut guillotine gates into measuring pockets, each of which holds one skipload of ore.

As previously noted, the use of measuring pockets or hoppers eliminates delays in loading and thus shortens the time of the hoisting cycle. Whereas 25 years ago comparatively few pockets were equipped with these auxiliary measuring hoppers, their use has become almost universal where rapid hoisting is required. The hoppers are filled while the hoist is in motion, and as soon as an empty skip comes to rest under one of them the loader opens the hopper gate, filling the skip. Since the hoppers hold only enough for a skipload, spilling is reduced to a minimum. When the skip is loaded directly from the large pocket, delays may be occasioned by hanging up of the ore back of the gate, and runs of fine material caused by inability to close the gate quickly may overflow the skip.

Skip pockets and measuring hoppers are of varied design, from simple ones such as that shown in figure 79, *B*, to very elaborate ones. They should be designed to suit the tonnage and nature of the material to be handled, and the general lay-out must conform in individual cases to the position of the haulageway relative to that of the shaft, the available depth below the level, and the nature of the rock and the way it stands. Interference with pump rooms, sumps, and discharge columns must be avoided; underground crusher stations and other factors may further complicate the problem.

In some mines where the ore breaks large, loading it into skips may be facilitated considerably by installing a primary crusher ahead of the pocket. This practice is common in Ontario and Quebec and not only speeds loading by preparing the ore so that it will run freely but also reduces wear on the pocket and skip linings and breakage of gates. Figure 75, *B*, shows a general lay-out where a main ore-pass system delivering ore to crusher stations at four separate horizons in the shaft is employed in conjunction with skip pockets and measuring hoppers.

Figure 80 shows a somewhat elaborate installation below a 500-ton main pocket. The main or upper gate is of the vertical sliding undercut plate type and is actuated by a large air cylinder. The ore is drawn through this gate into a measuring hopper equipped with an undercut arc gate actuated by another air cylinder. Below this gate is a short, steep slide or chute that directs the ore into a 112-cubic-foot skip.

It will be noted that the measuring hopper virtually has a bottom discharge. This installation was made at the United Verde mine.[23] The average time required for loading the 112-cubic-foot skip was 7 seconds, and the spill from hoisting 2,000 tons per day was 6 tons. Only one skip loader was required to handle this tonnage.

With some very sticky ores it is necessary to have a bottom discharge similar to those shown in figures 80 and 81, *A*, to prevent the ore from packing in the hopper.

With coarse, dry ore that does not pack, a front discharge such as that shown in figure 81, *B*, may be satisfactory. In this installation storage is provided in the raise at the right. The ore breaks in large chunks and the chute is provided with a finger gate. The measuring pocket is discharged by raising the vertical guillotine gate. This arrangement is very compact and requires only a small amount of excavation around the shaft.

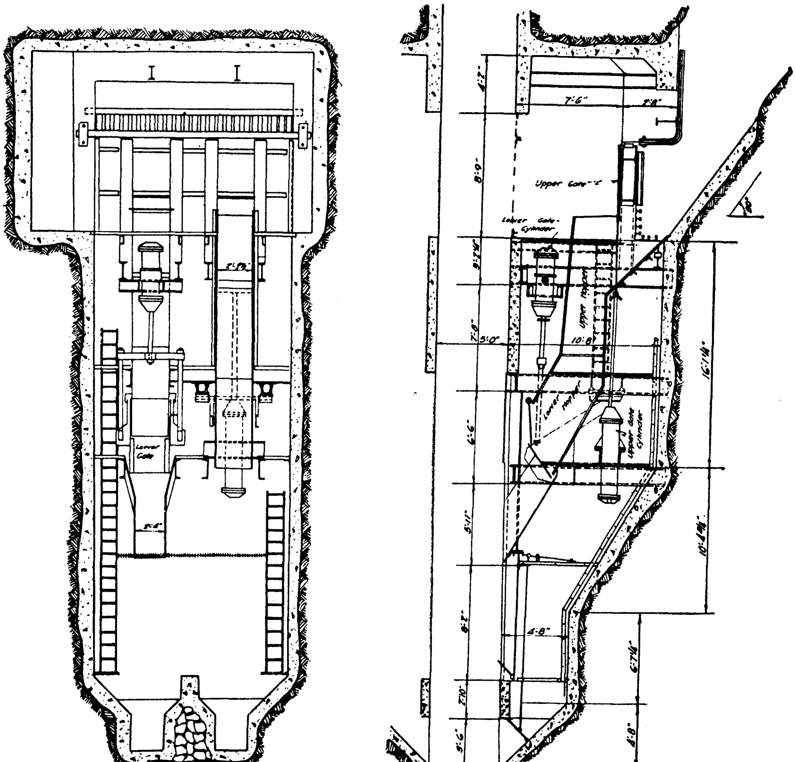


FIGURE 80.—Skip pocket at United Verde mine, Ariz., showing measuring hoppers.

The illustrations show only a few of many skip-pocket installations described in Bureau of Mines circulars and in the technical press but serve to indicate the various types in common use. At some mines the measuring pocket is a weigh-hopper, which, when filled, automatically closes the main pocket gate or stops a belt or pan feeder extending from the main pocket to the hopper. The discharge gate of the measuring pocket at some mines is opened automatically when the descending empty skip reaches the loading position, and at others the entire operation is automatic—opening and closing the main gate, filling the measure, and opening and closing the gate on the measure.

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[23] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, pp. 20–21.

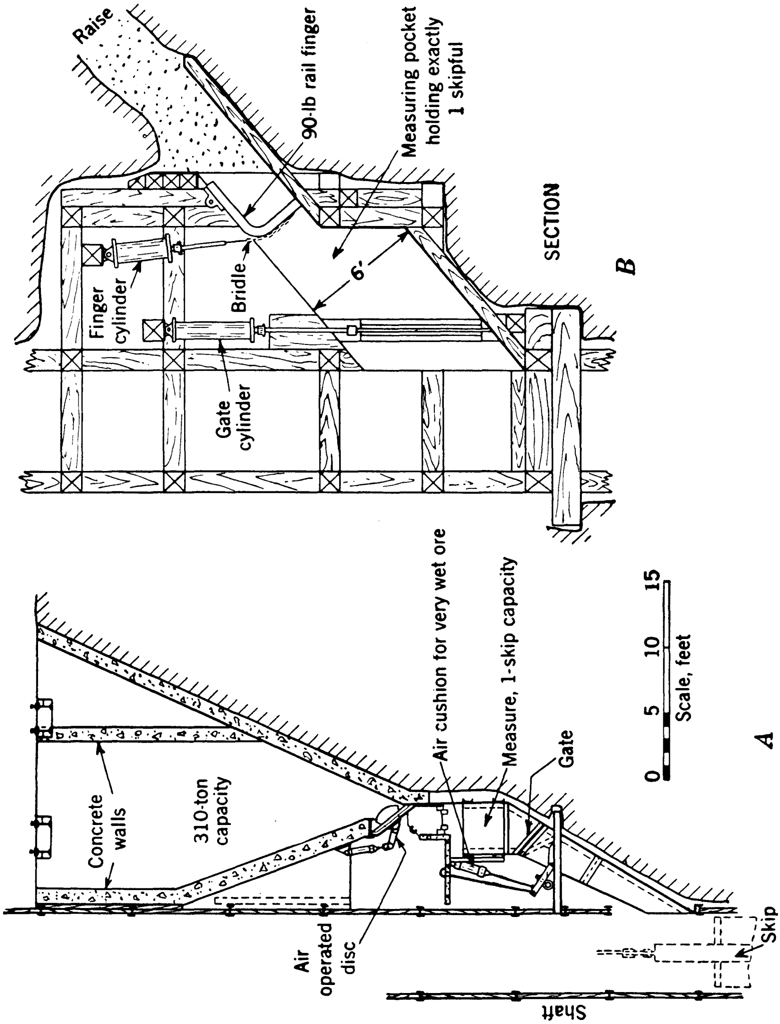
Raise
90-lb rail finger
Measuring pocket holding exactly 1 skipful
Finger cylinder
Bridle
6'
Gate cylinder
SECTION
B
Air cushion for very wet ore
Measure, 1-skip capacity
Gate
0 5 10 15
Scale, feet
A
Concrete walls
310-ton capacity
Air operated disc
Skip
Shaft

FIGURE 81.—*A*, Loading pocket with bottom-discharge measuring hoppers. *B*, Front-discharge measuringhopper.

COST OF STATIONS AND POCKETS

There are few published data covering the cost of cutting and timbering shaft stations and pockets. Except, perhaps, for the simplest lay-outs, such as those for caging cars, they are not standardized and vary widely as to size, nature of support (whether natural or of timber, steel, or concrete), ground conditions, type of gates and methods of operating them, and types of measuring hoppers and details of their construction and operation.

The excavations may involve raising, drifting, crosscutting, stoping, and sinking, and various methods for removing and disposing of the broken rock may be employed. Broadly speaking, the cost of actual excavation is roughly the same as for similar operations in other mine developments and in stoping, although work in close proximity to the shaft may cost somewhat more owing to the care required in blasting to avoid injury to the shaft. Thus drifting, crosscutting, raising, and sinking costs would approximate those of driving openings of the same size, which have been discussed under "Development." The cost of enlarging smaller openings to form shaft stations and ore pockets may be roughly the same on a tonnage or yardage basis as for stoping in similar ground, except that support of the ground must be of a permanent nature and therefore may cost considerably more. Thus it may be necessary to install heavy timbers or structural-steel supports and concrete linings or to gunite the walls.

A few data on costs of excavating stations and pockets are given below. At the Sylvanite mine, Kirkland Lake, Ontario, costs of cutting stations only on levels in 1931 were as follows.[24] The rock was syenite and syenite porphyry.

Total excavation, 4 levels, 22,358 cubic feet[1]

	Breaking	Hoisting	Timbering	Other costs	Total	Cost per cu. ft.
Material	$925	$263	$272	$114	$1,574	$0.070
Labor	2,591	1,030	224	292	4,137	.185
Overhead	941	1,362		510	2,813	.126
Total	4,457	2,655	496	916	8,524	.381
Per cubic foot	0.199	0.119	0.022	0.041	0.381	

[1] From 1,703 to 9,000 cubic feet per station.

The cost of cutting two stations comprising 8,196 cubic feet of excavation at the Vipond mine, Timmins, Ontario, was $2,885.49, or $0.35 per cubic foot. The rock was a basaltic schist or greenstone.

Costs of level stations, pump rooms, ore pockets, and skip-loading stations and equipment at Matahambre (Cuba) No. 2 shaft in 1930 were as follows:[25]

Costs per cubic yard[1]

	Cubic yards	Labor	Supplies	Power	Total
Level stations, pump stations, and sumps	2,315.6	$11.73	$7.90	$0.90	$20.53
Ore pockets, skip-loading stations, and equipment	1,156	20.60	21.01	1.94	43.55
Total and averages	3,471.6	14.69	12.27	1.24	28.20

[1] Costs include excavation, timbering, concreting sumps and ore pockets, lining ore pockets with rails, gates, measuring pockets, and structural steel for skip-loading equipment.

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[24] Jackson, Charles F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, pp. 52–53.

[25] Homer, Dudley D., Sinking and Equipment of the No. 2 Shaft at Minas de Matahambre: Explosives Eng., vol. 11, No. 2, February 1933, pp. 38–62.

HOISTING

The following discussion is restricted to a consideration of hoisting as the last phase of underground transportation and its relationship to the other operations involved in the mining of ore. The design and details of construction of hoists, cages, skips, and head frames involves the application of mechanical and electrical engineering principles that are not within the scope of this bulletin.

The mine operator, however, must specify the capacity of the hoisting equipment and limiting conditions, such as size of shaft compartments, tonnage to be handled, percentage of working time available for hoisting ore and waste, number and depths of levels from which hoisting is to be done, method of caging cars or loading skips, and requirements for safety in operation. He is concerned primarily with the cost of the equipment and its capacity for doing the work required of it with maximum efficiency and reliability and with minimum operating cost. Maximum efficiency can be obtained only if ore-breaking, loading, hauling, and hoisting be coordinated. Since the shaft is the bottle neck of the mine, it is preferable to have extra hoisting capacity to permit picking up of tonnage after unavoidable hoisting delays.

Reference has previously been made to buckets, cages, and skips, the operation of loading pockets, and the various methods of hoisting as they affect the shaft-station lay-outs. The advantages of skiphoisting, except for small-scale operations or under certain special conditions, have been pointed out.

For caging cars, landing chairs usually are provided for the cages to rest upon, thus permitting accurate spotting of the cage to match the elevation of the station tracks and eliminating sudden strains on the hoisting rope when the cars are run onto the cage. An alternative for chairs is the use of chains fastened to the station timbers on each side of the cage compartment and provided with a ring on the free end, which the station attendant or cager slips under hooks on the cage as it is lowered into landing position at the level. Gates on the ends of the cage usually suffice to hold the car on the cage during hoisting in the case of small cages and cars. With large cages and cars it is preferable to have automatic horns or stops on the bottom of the cage to hold the car in position.

Modern ore skips usually are of the self-dumping type and in vertical shafts are overturned in the head frame by means of dump plates (fig. 82). In this illustration, a cage is shown suspended below the skip for handling men and supplies. Skips for inclined shafts usually are built with four wheels, the rear pair of which have a wider tread than the front pair. When the skip approaches the dump position, the rear wheels ride up on one pair of rails set at a wide gage and steep inclination, the front wheels drop between them and run on a narrower-gage track set at a flatter inclination, and the skip is thus tipped into the dumping position. On very flat inclines, the mine cars may be hoisted directly on regular tracks.

Skips usually are constructed of steel, but in recent years the use of aluminum-alloy skips has increased in deep shafts where large tonnages are handled so as to reduce the tare load and correspondingly increase the ore load that can be handled without altering the power of the hoist. Skip capacity usually can be increased or decreased within reasonable limits simply by varying the length of the skip and height of the head frame, whereas the size of the cage compartment will limit the capacity of the car that can be employed with cage hoisting.

Usually ore is hoisted in balance, especially if the tonnage handled is large and the shaft deep. For this purpose the shaft must have two hoisting compartments with a skip (or cage) running in each and operated by the same hoist; as one rope winds on over the top of the hoist drum, the other unwinds underneath the drum, or vice versa. The weights of the two skips (or cages and cars) are thus in balance and the net load to be hoisted is only that of ore plus the difference in the weights of the suspended ropes in the two compartments. This difference in rope load is positive when the loaded skip is below the empty one, zero when the two skips are at the same elevation in the shaft, and negative after the midpoint in the hoisting cycle is passed. If hoisting is done from one level only, a single drum or two drums keyed to the same shaft may be used. If hoisting is done from several levels, one drum may be fixed on the shaft and the other clutched thereto to permit equalizing the rope lengths when hoisting is done from intermediate levels. Sometimes a skip is run in one compartment in balance with a cage in the other; or, if the output is small and the shaft has only one hoisting compartment, a single skip or cage may be balanced by a counterweight running in a division of the pipe and manway compartment or in a corner of the hoist compartment.

Hoist drums may be cylindrical, conical, or cylindro-conical, the first being the most popular because of its lighter weight, short length, lower first cost, simplicity, and adaptability to hoisting from several levels. Also, the rope may be wound over on itself in two or even more layers if necessary.

In very deep shafts and for heavy loads, however, the power required to accelerate the load on a large drum diameter is very great compared to the average power input during the hoisting cycle. To overcome this, conical drums have been employed, whereby the load is picked up on the small end of the cone and is accelerated gradually as the rope winds on a larger diameter at each turn. Conical drums are heavy, must be grooved, are large in diameter, and wide, and the rope cannot be wound past the center of the face. They are not well adapted to hoisting from several levels.

Conical drums have been superseded largely by cylindro-conical drums for deep hoisting from a single level. These drums are conical at the ends only, and the load is accelerated with the rope on the conical portion, and winding is finished on the cylindrical portion. They thus have all the advantages of conical drums and minimize their disadvantages, since they may be lighter and shorter and the ropes can be wound past the center. However, they are not well suited for hoisting from a number of different levels.

Power for operation of mine hoists is an extensive subject by itself and one that cannot be discussed in detail in this bulletin. Mine hoists are commonly driven by electric, steam, compressed-air, or gasoline power and each has its particular field, although the advantages of electric power favor its adoption where it is available, reliable, and its cost is not prohibitive. An electric hoist does not consume power or fuel when not actually in operation; accurate automatic acceleration, speed, braking, and overwind control can be provided readily, and first cost and maintenance charges are comparatively low. Electric hoists may be driven by either A. C. or D. C. motors.

The first cost of an A. C. installation is considerably lower than that of a D. C. installation and it is often satisfactory for small hoists and for hoisting heavy loads from deep shafts, especially when hoisting is done from only one level, but from the standpoint of speed

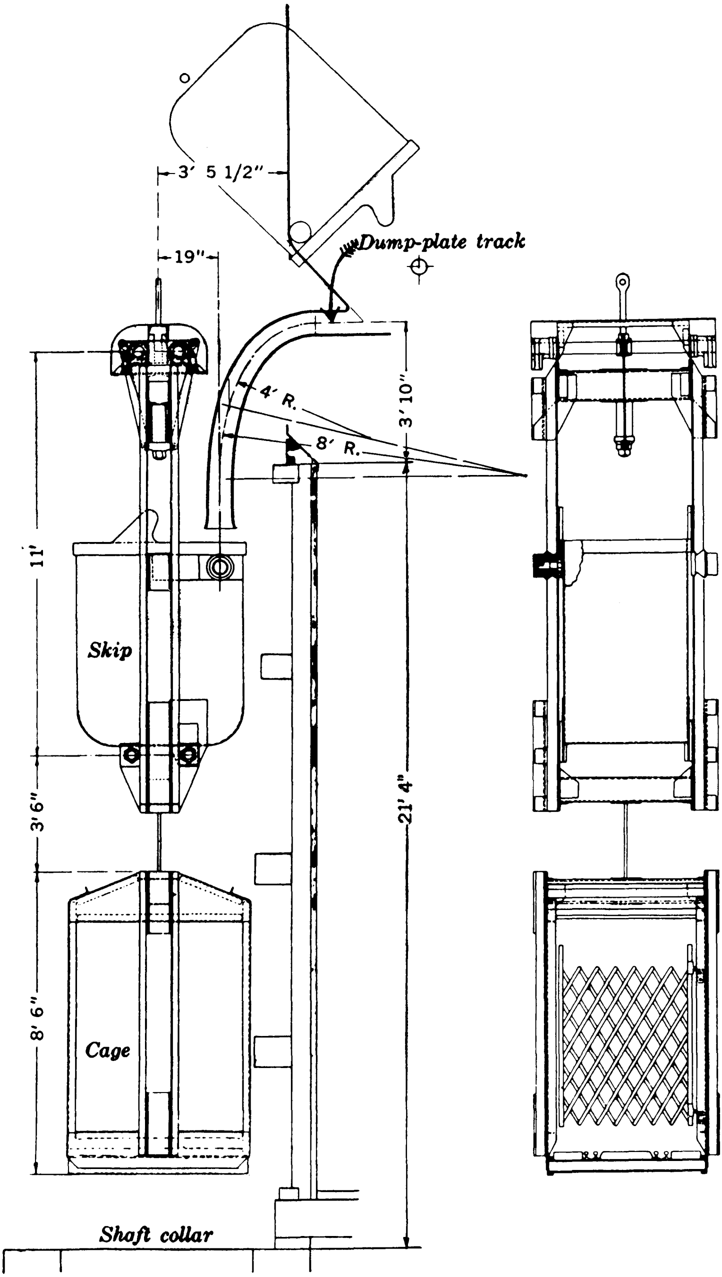
3′ 5 1/2″
19″
Dump-plate track
4′ R.
8′ R.
3′ 10″
11′
Skip
3′ 6″
21′ 4″
8′ 6″
Cage
Shaft collar

FIGURE 82.—Skip, cage, and dump plate for vertical shaft.

TABLE 32.—*Hoisting data*

Mine	Depth of shaft, feet	Number of hoisting levels	Hoisting method	Ore load per trip, pounds	Ore hoisted per 8-hr. shift, tons	Hoisting speed, feet per minute	Method of loading	Hoist		Power, current, and motor rating	Control and safety devices	Cost of hoisting, per ton
								Type of drum	Dimensions of drum, inches [1]			
Hartley-Grantham, Tri-State dist.[2]	200	1	Cans	1,200	[3] 400±	1,200	Hook to rope.	Cyl.; S. D.		75-h. p. D. C.		[4] $0.03 to $0.07
Hartley, Tri-State dist.[5]	350	1	do	1,350	570	1,200±	do	do	15 by 24	Diesel, 120 h. p.		
Burra-Burra, Tenn.[6]	1,600 (70°)	13	Skips balanced	14,000	775	700	Direct from car or pocket.	Cyl.; D. D.	72 by 48	400 h. p. A. C., 2,300 V.		
No. 8, Southeast Missouri.[7]	618	1	do	8,000	500±	750	Direct from pocket.	S. D.; Cyl.	72 by 66	250 A. C., 2,200 V.	Lilly	.0445
Mascot, Tenn.[8]	582	1	do	10,000	1,150±		do					.043
Engels, Calif.[9]	830	5	do	4,000	450	800	do			150 h. p. electric		
Eighty-Five, N. Mex.[10]	1,650	13	do	6,000		1,000	Measuring hopper.			Electric		.24
Verde Central, Ariz.[11]	1,930	10	Double-deck cages balanced.	2,000	300	1,100	Caging			D. C.		
Eustis, Quebec, main shaft.[12]	6,350 (45°±)	1	Skips, balanced	7,000	71,709 (1 yr.)	750	Measuring hopper.	D. D.; Cyl.	96 by 48	250 h. p.	Lilly	.2668
Mt. Hope, N. J., main shaft.[13]	1,200 (64°)	1	do	6,000	500±	700	do			Two 125-h. p. D. C. motors.	do	
Hog Mountain, Ala.[14]	215	1	Cage, unbalanced.	2,000	68		Caging	S. D.; Cyl.		50-h. p., 220 V. D. C.		.127
Matahambre, Cuba.[15]	2,000 (ultimate 4,000)	2	Skips, balanced	10,000	1,000	1,000	Measuring hopper.	D. D.; Cyl.	120 by 78	508-h. p. av., 450 V., D. C.[16]	W. L. I., Lilly	
		(?)	do	8,000	6,600	1,600		do	do	765-h. p. av.	T. L. S.	
Pilares, Mexico [17]	[18] 2,200	3	do	8,000		1,200	do	do	108 by (?)	400-h. p. D. C., 300 V.		
United Verde, Ariz.[19]	[20] 2,400+	7?	do	12,000 to 16,000	2,000	890	do	do	120 by 60	650-h. p. 500 V., D. C.	W. L. I.	
McIntyre, Ontario.[21]	4,125	1	do	12,000	1,000±	3,000	do	do	144 by 96	2,450-h. p. D. C.	W. L. I. Whitmore overwind and overspeed.	

See footnotes at end of table.

TABLE 32.—*Hoisting data*—Continued

Mine	Depth of shaft, feet	Number of hoisting levels	Hoisting method	Ore load per trip, pounds	Ore hoisted per 8-hr. shift, tons	Hoisting speed, feet per minute	Method of loading	Hoist		Power, current, and motor rating	Control and safety devices	Cost of hoisting, per ton
								Type of drum	Dimensions of drum, inches			
Hecla, Idaho [22]	[20] 2,800	7?	Skips, balanced	14,000	500±	2,400	Direct from pocket.	D. D	120 by 72	2,100-h. p. D. C., 500 V.	W. L. I	
Argonaut, Calif.[23]	5,800 (70°)	5 or more.	do	7,100	130±	900	do	Cyl.; D. D	96 by 47	500-h. p. A. C	M. S. & L. R., O. W. and O. S.	$0.519
Butte and Superior, Mont.[24]	3,700	Several	do	14,000	250±	2,300	do	do	108 by 94	2,000-h. p. 500 V., D. C.	Welch O. W. and O. S.	
Eureka Standard, Utah.[25]	1,424	5	Skips or cages, balanced.[26]	4,000 (skip) 2,000 (cage)	200 (ore and waste).		do	do	42 by 54	200-h. p. A. C		.490
Inspiration, Ariz.[27]	630	2	Skips in balance.[28]	25,000	[29] 3,800, [30] 5,300	750	Automatic measuring hopper.	do	120 by 65	585-h. p. D. C	W. L. I., hand or automatic.	
Miami, Ariz.[31]	811	1	do	21,000	7,500± capacity.	2,250		do		Two 1,400-h. p. D. C. motors.	W. L. I	.033
Mesabi, Minn.[32]	256	1	do	6,000	400±	1,000	Direct from pocket.	Cyl.; S. D	60 by 72	50-h. p. A. C	Automatic	
Montreal No. 5, Wis.[33]	3,000; 2,300 av.	4	do	17,920	1,800	1,850	Measuring hoppers.	Cyl.; D. D	96 by 96	2,300-h. p. D. C	W. L	[34] .158
Mt. Isa, Queensland.[35]	700 (No. 4 level); 875 (No. 5 level).	2, mostly from No. 4.	do	18,000	2,250 to 2,700 from No. 4.	1,890	do	do		1,525-h. p. D. C	W. L. manual or remote.	
El Potosi, Mexico:[36]												
No. 1 shaft	2,777		do	8,800	675±	1,500	do	do	96 by 78	700-h. p. D. C	W. L. I.; L. S. Lilly.	.08
No. 5 shaft	2,587		do	6,600	525±	1,200	do	do	84 by 72	400-h. p. D. C	do	
Hollinger, Ontario,[37] (2 duplicate hoists.)	3,125	1	do	12,000	1,800 capacity, 1 hoist.	3,000	do	do	120 by 144	2,000- h. p. A. C	Lilly O. W., O. S.	.168
Frood No. 3, Ontario.[38]	3,045	1	do	22,000		3,000	do	do	144 by 84	3,200-h. p. D. C	W. L. I	
Frood No. 4, Ontario.	3,645±	1	do	20,000		2,000	do	do	144 by 96	2,000-h. p. D. C	do	

[1] Diameter by face.

[2] Keener, O. W., Methods and Costs of Mining at the Hartley-Grantham Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6286, 1930, 8 pp.

[3] It is reliably reported that 150 cans, or 90 tons per hour, can be hoisted by this system from depths of 250 to 300 feet.

[4] Average reported cost for all mines of the Tri-State district hoisting in cans.

[5] Anderson, C. N., Mining Methods and Costs at the Interstate Zinc and Lead Co.'s Hartley Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6656, 1932, 16 pp.

[6] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[7] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

[8] Coy, H. A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Bureau of Mines Inf. Circ. 6239, 1930, 11 pp.

[9] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[10] Youtz, R. B., Mining Methods at the Eighty-Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.: Bureau of Mines Inf. Circ. 6413, 1931, 26 pp.

[11] Dickson, R. H., Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.: Bureau of Mines Inf. Circ. 6464, 1931, 13 pp.

[12] Snow, Fred W., and Brownbill, H. F., Mining Methods and Costs at the Eustis Mine: Canadian Min. and Met. Bull. 286, February 1936, Transactions Sec., pp. 70–85.

[13] Sweet, J. R., Mining Methods and Costs at the Mount Hope Mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp.

[14] Johnson, N. O., Mining and Milling Methods and Costs, Hog Mountain Gold Mining & Milling Co., Alexander City, Ala.: Bureau of Mines Inf. Circ. 6914, 1936, 23 pp.

[15] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

[16] Horsepower given is average input to M. G. set.

[17] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares, Sonora, Mexico: Bureau of Mines Inf. Circ. 6307, 1930, 34 pp.

[18] Depth limit of hoist, 2,500 feet.

[19] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[20] Below haulage-tunnel level.

[21] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, 19 pp.

[22] Foreman, C. H., Mining Methods and Costs at the Hecla and Star Mines, Idaho: Bureau of Mines Inf. Circ. 6232, 1930, 21 pp.

[23] Vanderburg, W. O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6311, 1930, 14 pp.

[24] McGilvra, D. B., and Healy, A. J., Methods and Cost of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Bureau of Mines Inf. Circ. 6370, 1930, 16 pp.

[25] Gardner, E. D., Mining Methods and Costs at the Eureka Standard Mine: Bureau of Mines Inf. Circ. 6851, 1935, 14 pp.

[26] Can hoist with 2 skips in balance, 1 skip and 1 cage in balance, or 2 cages in balance. Cages used for transferring waste between levels.

[27] Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

[28] 2 duplicate hoists.

[29] 1 hoist using automatic control.

[30] 1 hoist using manual control from loading floor.

[31] Maclennan, F. W., Miami Copper Co. Method of Mining Low-Grade Ore Body: Am. Inst. Min. and Met. Eng. Tech. Pub. 314, 44 pp.; *also* Trans. Am. Inst. Min. and Met. Eng., vol. 91, 1930 (Yearbook), pp. 39–86.

[32] Haselton, W. D., Underground Mining Practice and Costs at a Mesabi Range (Minn.) Mine Using the Top-Slicing System: Bureau of Mines Inf. Circ. 6325, 1930, 11 pp.

[33] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

[34] Includes also cost at 1 inclined shaft.

[35] Kruttschnitt, J., and Mann, V. I., Mining Methods and Costs at the Mount Isa Mines, Ltd., Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

[36] Walker, Harlan A., Mining Methods and Costs at El Potosi Mine, El Potosi Mining Co., Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6804, 1934, 38 pp.

[37] Canadian Min. Jour., September 1935, pp. 391–395.

[38] Canadian Min. Jour., vol. 58, No. 11, 1937, pp. 605–607.

NOTE.—Abbreviations used in table are as follows:

Cyl.=Cylindrical.
S. D.=Single drum.
D. D.=Double drum.
D. C.=Direct current.
A. C.=Alternating current.
V.=Volts.
W. L.=Ward-Leonard.
I.=Ilgner fly-wheel set.
T. L. S.=Track limit switches.
O. W.=Overwind device.
O. S.=Overspeed device.
M. S.=Master switch.
L. R.=Liquid rheostat.

control it is not as satisfactory as D. C. installations. Large-capacity, high-speed, D. C. hoists usually are provided with Ward-Leonard control, and braking is electrical, although mechanical brakes are employed for locking the hoist at the end of the trip. In the Ward-Leonard system, a motor-generator set driven by an induction motor supplies D. C. current to the hoist motor. The generator of the set and that of the hoist motor are excited separately. The generator field is reversible and can be varied from zero to full voltage in either direction by resistances that cause the pressure and current of the hoist motor to vary, likewise, from zero to maximum. In the Ward-Leonard-Ilgner system a heavy flywheel is added to the motor-generator set to equalize the power demand during the hoisting cycle.

Gas-engine drive for mine hoists is restricted to small installations in remote districts where electric energy is not available and the cost of solid fuel for steam production is excessive. Where fuel oil could be used for making steam it probably would be preferable to use oil in Diesel engines for producing electrical energy. Small gasoline hoists, however, have their place and often are the most economical installation for prospecting and other small-scale work. Compressed-air hoists are very uneconomical in the use of power but are sometimes useful as a temporary expedient for short-lived underground installations, and for sinking operations pending the installation of a permanent hoist.

Table 32 presents data on a number of typical hoisting installations of various types and capacities.

The hoists mentioned in table 32, with one exception, are operated electrically. Elsing [26] has given some data on hoisting with steam at the Sacramento shaft, Bisbee, Ariz. The shaft is vertical and the total cost of hoisting by steam during 1914 and 4 months of 1915 was 7.1 cents and 6.9 cents, respectively, per ton of ore, and comprised the following items:

Item	Cents per ton hoisted	
	1914	4 months 1915
Steam [1]	1. 9	2. 3
Electricity	. 1	-------
Engineers	. 9	1. 0
Oilers	. 5	. 5
Skip loaders	1. 2	1. 0
Miscellaneous	. 1	. 4
Hoist repairs	1. 3	. 6
Cage repairs	. 4	. 3
Ore pockets	. 2	. 5
Ropes	. 5	. 3
Total	7. 1	6. 9

[1] Cost of steam at the boiler house was 45 cents per 1,000 pounds.

At the Montreal mine,[27] hoisting at the No. 4 shaft was done with steam in 1928. The lowest level was about 2,700, measured along the

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[26] Elsing, Morris J., Cost of Hoisting Lowest at Porphyry Mines: Eng. and Min. Jour., vol. 135, No. 2, February 1934, pp. 59–62.

[27] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

62½° inclined shaft. Hoisting was in balance with skips of 5-ton capacity. During 1928, cost of hoisting 450,000 long tons was $0.189 per ton.

TRANSPORTATION OF WASTE

The handling of waste through waste passes and its transportation in cars on the levels and in shafts have been mentioned in the foregoing pages. Methods of handling waste depend on its source and whether the waste is to be utilized as filling in stopes for support of the walls, or is merely to be disposed of as cheaply and conveniently as possible.

Waste rock from development work may be sent to the surface and disposed of there or may be used as stope filling. If possible to do so, it is often cheaper to dump it in worked-out open stopes than to hoist it, even though it may not be required for ground support. In any event, its handling and transportation are sometimes a major problem.

It is often necessary, where filled-stope methods of mining are employed, to obtain additional waste from outside sources, such as waste stopes, surface quarries, or gravel pits, which introduces another transportation problem. The handling of waste and other filling material in the stopes themselves will be discussed later under Stoping Methods.

Development rock must be distributed in cars from scattered collecting points to the tops of the stopes where it is required and in filled-stope methods of mining it must be transported when and in the quantities needed to avoid delaying ore production. Unless work is well organized the transportation of waste will interfere with hauling and hoisting of ore. The use of waste storage raises or pockets has been mentioned already as an aid to avoidance of this trouble, and, as previously indicated, it is sometimes advisable to drive separate waste and supply drifts in the hanging wall to separate ore and waste tramming operations.

Development rock frequently must be hoisted from a lower to a higher level in the mine; usually, deep mines are developed and worked from the top down, with most of the stoping above the levels where the principal development work is being done. Rock from the lower levels may be hoisted in the cars on cages and taken off on the level where it is to be dumped. Where only skips are available for hoisting, there are three alternatives: (1) Hoist to surface and dispose of it there; (2) hoist to surface and dump into a main waste pass, whence it can be drawn off into cars at any desired level; (3) provide dump plates in the shaft and waste pockets at intermediate levels, where it can be drawn off into cars as needed. The second is the preferred alternative if the rock is needed for stope fill, and the third may be employed where the rock is needed for fill, though it is not very satisfactory and rarely is used.

Where waste filling is procured from surface glory holes, tailing piles, or gravel pits, main waste passes are to be preferred. These may be raises to surface, as previously discussed, or large bore holes. The practice at Matahambre, Cuba, where sands are settled out from the mill tailing and piped with water down raises through rubber-lined pipe and thence to the stopes is reported to be very successful.[28]

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[28] Richert, George L., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp. Filling Stopes with Mill Tailing: Eng. and Min. Jour., vol. 127, No. 9, Mar. 2, 1929, p. 348. Homer, Dudley D., Rubber Pipe Lining Minimizes Pulp Abrasion: Eng. and Min. Jour., vol. 132, No. 8, Oct. 26, 1931, pp. 367–368.

If ore and waste must be handled in the same shaft, particularly if on the same cages or skips, waste hoisting obviously interferes with ore hoisting, and vice versa. If the capacity of the hoisting equipment is small, it is apparent that this condition may be very serious. It is the practice at some mines where there is more than one hoisting shaft to reserve one of them exclusively for hoisting ore. At other mines ore is hoisted in skips and waste in cars on cages in separate compartments of the same shaft.

UNDERGROUND TRANSPORTATION OF EQUIPMENT, MATERIALS, AND SUPPLIES

It is essential to efficient operation that equipment, materials, and supplies required in the conduct of mining operations be handled with dispatch so that they may be available in the quantities required when and where needed. If the supply of timber or sharp drill steel, pipe fittings, or other materials convenient to the working place is inadequate for needs, it is often impossible to complete a round in a stope or heading during the shift. On the other hand, storage of too great a supply of materials underground where they may deteriorate rapidly and will tie up capital unnecessarily is to be avoided.

A properly stocked warehouse on surface with facilities for economical transportation of supplies to strategically situated distributing points underground will do much to increase general efficiency. Where steel, explosives, timber, pipe, and pipe fittings are stocked in small quantities underground they should be stored at places convenient of access by the miners so that a minimum of time may be lost in removing them to the place where they are to be used. From experience in mine operation and observations in hundreds of mines, it is evident to the authors that a vast amount of time and energy is wasted by miners in gathering supplies, which could be spent more effectively at the working face.

The principal items of supplies and equipment that must be handled are timber, drill steel, explosives, pipe and fittings, rail, rock drills, drilling gear, and mountings.

In mines employing square-set, top-slicing, stringer-set-and-fill, or sublevel-caving methods of mining it is necessary to handle large quantities of timber. Even where other methods are employed, considerable timber may be required for timbered haulageways, chute construction, cribbed or timbered ore and waste passes, and stull supports in stopes.

Some mines in hard or abrasive rock use large quantities of drill steel, and it is often a problem to take in sharp steel and bring out dull steel without serious interference with the tramming and hoisting of ore. The handling of explosives is a special problem, requiring as it does observance of safety precautions and restrictions on the quantity that can be handled and lawfully stored in any one place.

Many large mines make it a practice to handle timber, drill steel, and explosives, as well as other supplies used in large quantities, between the regular shifts when the shaft and haulageways are otherwise free. This, of course, entails the employment of a special crew for this work, which may not always be economical in small mines where this crew is not kept busy during its entire shift.

At shaft operations much time can be saved by providing a long cage in one compartment upon which timber, steel, and powder trucks can be run and taken off at the various stations without the necessity of rehandling between the timber yard, shop, or powder magazine on surface and the place of delivery underground. In adit

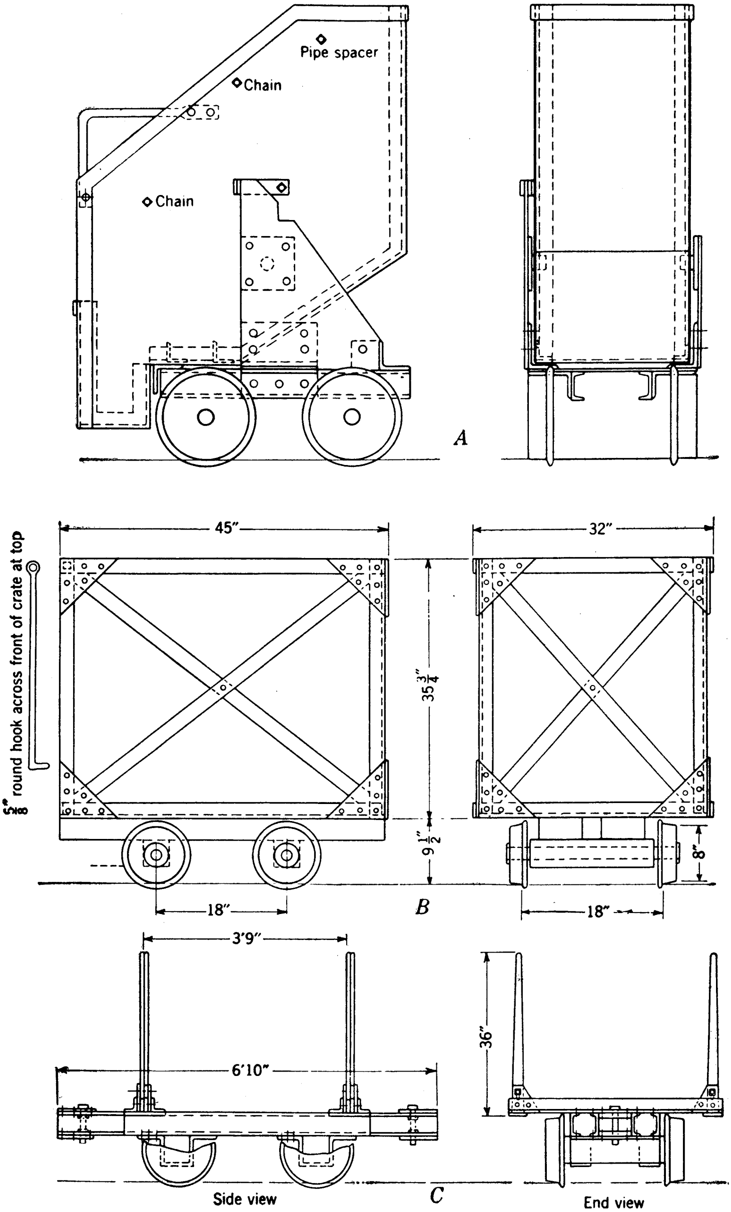
Pipe spacer
Chain
Chain
A
5/8" round hook across front of crate at top
45"
35 3/4"
9 1/2"
18"
B
32"
8"
18"
3'9"
6'10"
36"
Side view
C
End view

FIGURE 83.—Supply trucks. *A*, Drill steel car for small cage. *B*, Timber crate. *C*, All-steel timber truck.

or tunnel operations suitable trucks for each type of supplies may be provided to facilitate handling. Where timber and steel must be handled on small cages or skips, much time is often consumed stacking it on end in the cage a few pieces at a time and taking it off again on the level. Some mines with small cages save considerable time by using a steel car in which all but the longest steels can be stood on end. (Fig. 83, *A*.) Cars of this type are not suitable for rapid transportation as they have a short wheel base and when loaded are top-heavy. Where the cage is long it is more convenient to rack the steel horizontally on a long truck provided with separate compartments for each length or gage of steel.

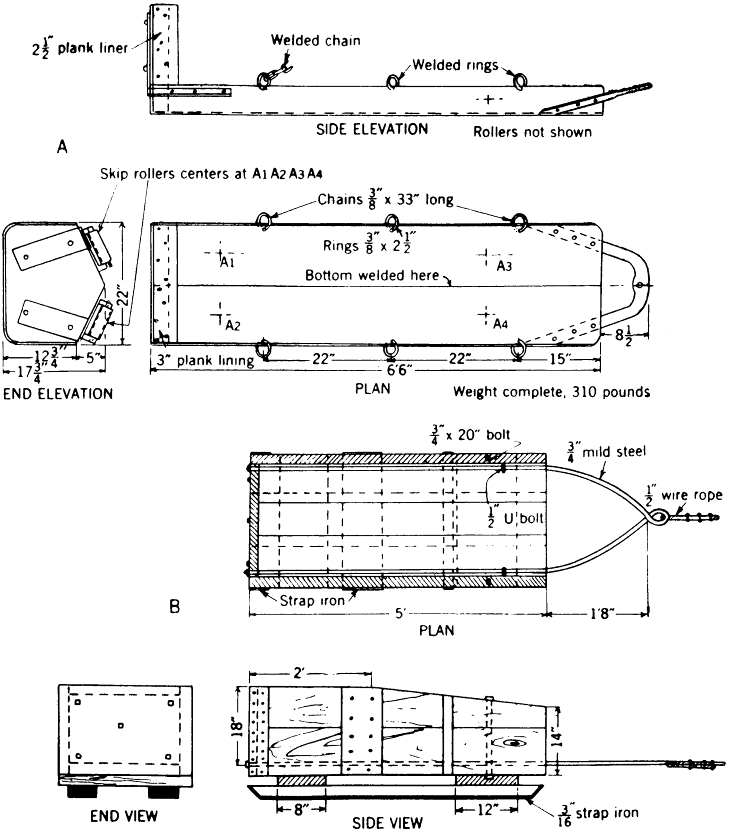
2½" plank liner
Welded chain
Welded rings
SIDE ELEVATION
Rollers not shown
A
Skip rollers centers at A1 A2 A3 A4
Chains ⅜" x 33" long
Rings ⅜" x 2½"
Bottom welded here
A1
A2
A3
A4
22"
12¾"
5"
17¾"
END ELEVATION
3" plank lining
22"
22"
15"
6'6"
8½
PLAN
Weight complete, 310 pounds
¾" x 20" bolt
¾" mild steel
½" wire rope
½" U bolt
Strap iron
B
5'
1'8"
PLAN
2'
18"
14"
8"
12"
3/16" strap iron
END VIEW
SIDE VIEW

FIGURE 84.—*A*, "Boat" for hoisting supplies into stopes. *B*, Timber cage for hoisting timbers into stopes.

Figure 83, *B*, shows a "crate" used for transportation of timber at the United Verde Extension mine, and figure 83, *C*, shows an all-steel timber truck with removable side stakes employed at the Montreal mine (Wisconsin).

The use of so-called tugger hoists for hoisting timber, drill steel, and other supplies into stopes through stope or service raises has become almost universal and saves a great deal of time and arduous labor. Figure 84, *A*, illustrates a "boat" employed at the Park-Utah mine for hoisting supplies into stopes. The boat operates in a

V-shaped trough slide made of 2-inch plank and is equipped with rollers to reduce the wear on the boat and slide. Figure 84, *B*, depicts a timber cage that slides on runners over a planked bottom in one side of the stope manway.

STOPING

The subject of stoping has been discussed at considerable length by Jackson and Gardner in a Bureau of Mines bulletin.[29] Data in this bulletin were compiled principally from those presented in reports, dealing with methods and costs at about 75 individual mines, previously published in information circulars. The principal mining methods have been summarized in a series of information circulars as follows:

Inf. Circ. 6193.—Mining Ore in Open Stopes, Central and Eastern United States, by Chas. F. Jackson. 1929. 36 pp.

Inf. Circ. 6293.—Shrinkage Stoping, by Chas. F. Jackson. 1930. 54 pp.

Inf. Circ. 6350.—Undercut Block-Caving Method of Mining in Western Copper Mines, by E. D. Gardner. 1930. 44 pp.

Inf. Circ. 6410.—Mining by the Top-Slicing Method with Some Notes on Sublevel Caving, by Chas. F. Jackson. 1931. 51 pp.

Inf. Circ. 6688.—Cut-and-Fill Stoping, by C. H. Johnson and E. D. Gardner. 1933. 58 pp.

Inf. Circ. 6691.—Square-Set System of Mining, by E. D. Gardner and William O. Vanderburg. 1933. 73 pp.

The following discussions of stoping have been condensed from the earlier bulletin, whereas the earlier cost data have been amplified by data from subsequent reports.

In this bulletin, the term "stoping" is employed in its broader sense to mean the operation of excavating ore by means of a series of horizontal, vertical, or inclined workings in veins or large, irregular bodies of ore or by rooms in flat deposits. It covers the breaking of ore and its removal from underground workings, except those driven for exploration and development, and the timbering or filling of the stopes for the purpose of support.

Ore deposits vary greatly in their physical characteristics and with respect to the economic problems involved in their exploitation. Various methods of stoping have been devised for extracting the ore safely and economically from deposits of different types, and a nomenclature has been developed for use in referring to the various methods. Some of the terms are descriptive and require no interpretation, whereas others, particularly those applied to variations of the principal methods, are not. Among the variations are those bearing the names of mines at which they were devised or the names of the originators.

Basically, the stoping method or methods that can be applied to a given ore body depend on the requirements for support of the stope—the maximum area or span of back and walls that will be self-supporting during the removal of the ore; the nature, size, and interval between supports required to maintain the backs and the walls of the excavations; and the requirements for permanently supporting the overlying and surrounding country rocks and overburden to prevent their movement and subsidence. Variations of the principal methods of stoping may be based upon the direction or angle of working, sequence of operations, or methods of handling the broken ore.

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[29] Jackson, Chas. F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, 296 pp.

STOPING METHODS

A classification of stoping methods based upon method of support was adopted by the Mining Division of the Bureau of Mines in 1928. This classification has been used ever since as a basis for describing methods in its publications dealing with mining methods and costs and is as follows:

Classification of stoping methods

A. Stopes naturally supported.
  1. Open stoping.
    (*a*) Open stopes in small ore bodies.
    (*b*) Sublevel stoping.
  2. Open stopes with pillar supports.
    (*a*) Casual pillars.
    (*b*) Room (or stope) and pillar (regular arrangement).
B. Stopes artificially supported.
  3. Shrinkage stoping.
    (*a*) With pillars.
    (*b*) Without pillars.
    (*c*) With subsequent waste filling.
  4. Cut-and-fill stoping.
  5. Stulled stopes in narrow veins.
  6. Square-set stoping.
C. Caved stopes.
  7. Caving (ore broken by induced caving).
    (*a*) Block caving; including caving to main levels and caving to chutes or branched raises.
    (*b*) Sublevel caving.
  8. Top slicing (mining under a mat that, together with caved capping, follows the mining downward in successive stages).
D. Combinations of supported and caved stopes. (As shrinkage stoping with pillar caving, cut-and-fill stoping with top slicing of pillars, etc.)

Forced caving (a method employed at several very large mines and therefore important) has heretofore been classed by the Bureau as a variation of shrinkage stoping. In reality it is intermediate between shrinkage (supported) stopes and caved stopes, as far as support is concerned. The ore is broken by large blasts, and considerable caving often follows the blasts, yet the operation is fundamentally quite different from natural caving induced by undercutting, as in block-caving or sublevel-caving systems.

A. STOPES NATURALLY SUPPORTED: OPEN STOPES

Stopes naturally supported are those in which no regular artificial method of support is employed, although occasional props, cribs, or stulls may be used to hold local patches of insecure ground. The walls and roof are self-supporting. The simplest form of open stope is that in which the entire ore body is removed from wall to wall without leaving any pillars (fig. 85). It is applicable to relatively small ore bodies, as there is a limit to the length of unsupported span that will stand without support even in the firmest and strongest rocks. In sublevel stoping the ore is excavated in open stopes, retreating from one end of the stope toward an entrance at the other end (fig. 86) on each of a series of sublevels that are usually 20 to 25 feet apart vertically but may be 40 feet or more apart.

In open stopes with pillar support, the length of unsupported span is reduced by leaving pillars of ore. These pillars may be of the irregular or "casual" type, their position and size being determined by localized ground conditions (fig. 87, *A*), or may be regular in size and arrangement, conforming to a predetermined pattern (fig. 87, *B*).

Open stoping is applicable to deposits of strong, firm ore having strong, firm walls. In the narrower deposits (30 to 50 feet wide) the ore often can be mined the full width in one operation without the use of pillars. In wide deposits it usually becomes necessary to leave solid pillars to reduce the length of unsupported span and thus prevent the failure of back or walls. When "casual" pillars are used, it frequently is possible to leave low-grade ore within the ore body as pillars, at least in part, and thus make possible more complete recovery of the higher-grade ore. In general, casual pillars are used in the firmest type of ground and in deposits of variable thickness where their size and spacing can be proportioned to suit the conditions of the back and the height of the ore. Often a somewhat higher percentage of the ore is recovered than where a regular room-and-pillar system is employed.

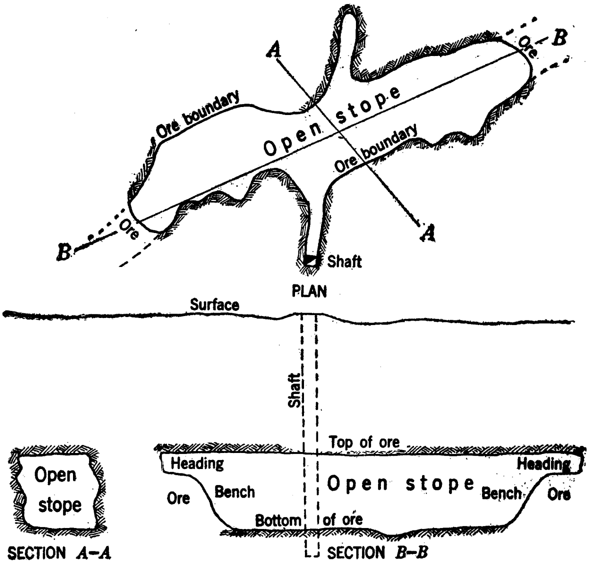
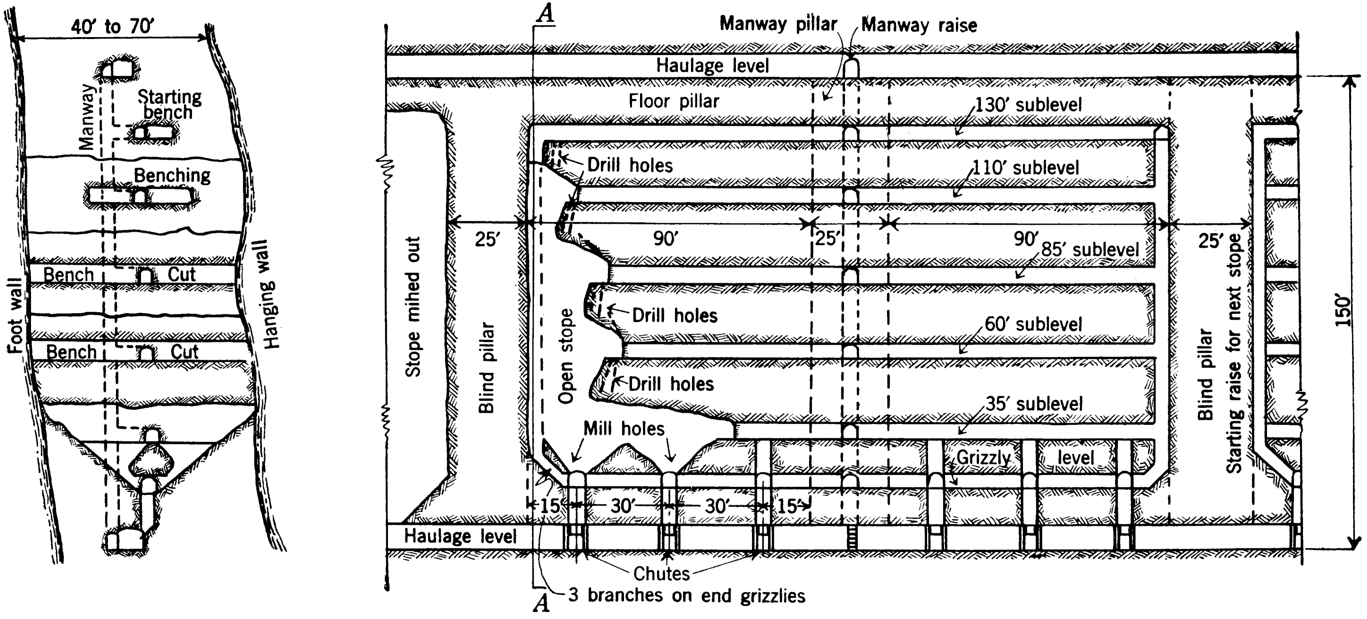


FIGURE 85.—Open stoping without pillars in small ore bodies.

A regular room-and-pillar pattern is applied chiefly to relatively thin, regular, flat-lying beds, where the roof must be definitely and permanently supported, in other words, where it is not safe to rely on personal judgment of the foreman or boss as to the amount of pillar support required at individual local points.

In the sublevel variation of open stoping, the stope faces retreat from the end of the ore body or stope limit longitudinally or (where the stopes are run across the lode) from one of its walls. The miners are always working under solid backs and have a safe avenue of retreat through the subdrifts to the manway raises. Slabbing of roof or walls at the worked-out end of the stope does not endanger the miners and, if the stope is worked back rapidly, usually will not cause serious dilution of the broken ore with waste. This system is applicable to


40′ to 70′
Manway
Starting bench
Benching
Bench
Cut
Bench
Cut
Foot wall
Hanging wall
A
Manway pillar
Manway raise
Haulage level
Floor pillar
130′ sublevel
Drill holes
110′ sublevel
25′
90′
25′
90′
85′ sublevel
25′
Stope mihed out
Blind pillar
Open stope
Drill holes
60′ sublevel
Drill holes
35′ sublevel
Mill holes
Grizzly
level
Blind pillar
Starting raise for next stope
150′
15′
30′
30′
15′
Haulage level
Chutes
3 branches on end grizzlies
A

VERTICAL PROJECTION *A–A*

VERTICAL LONGITUDINAL SECTION

FIGURE 86.—Typical sublevel stoping.

wide, thick, irregular masses of ore that stand well without artificial support and that have firm walls and to thick, tabular ore deposits that dip 50° or more (and, by using scrapers for removing broken ore, to flatter deposits). It may be used successfully in weaker ground than could be worked safely by other open-stope systems, as the miners are always protected by solid ground overhead. Large open stopes are mined by this method where the ore is so soft that it can be drilled with auger steel, and the method can be applied where the ore is too weak to be safely minable by shrinkage stoping.

B. STOPES ARTIFICIALLY SUPPORTED

1. SHRINKAGE STOPING

In shrinkage stoping the ore is mined in successive flat or inclined slices, working upward from the level or the bottom of the block of ore. After each slice or cut, enough broken ore is drawn off from below to provide a working space between the top of the pile of broken ore and the back of the stope (fig. 88). Usually about 35 to 40 percent of the ore will be drawn off during active mining in the stope. The remaining ore serves as a floor upon which to work in drilling the back for the succeeding cuts and also provides some support of a temporary nature to the stope walls. For this reason shrinkage stopes are considered to be a form of artificially supported stope, although undoubtedly there are good reasons why some consider them to be open stopes.

In narrow veins or lodes and those of moderate width, the stopes usually are run longitudinally and are mined from wall to wall without leaving pillars, except locally, to support bad ground or where the ore is too low-grade to be mined profitably. In wide ore, to reduce the unsupported span of the stope back, it may be necessary to mine the ore in a series of transverse stopes between intervening pillars of ore. The stopes then end against the walls of the vein and the sides are vertical pillars of ore.

After active mining has been completed to the level above or to the floor pillar, the rest of the broken ore is drawn off from below, leaving the stope empty. It may be filled with waste later to prevent general movement and subsidence or to permit mining of pillars left between stopes during the first mining.

Stulls may be employed during active mining to support local patches of insecure ground. Where the width is not too great, they often are employed when the stope is being emptied to protect the miners from falls of loose ground while cleaning down broken ore that has hung on the footwall and recovering ore that was left in the walls during stoping. Stulls 30 to 40 feet long have been used for this purpose,[30] but the safety and adequacy of this type of support are questionable for widths as great as this. While stulling from the top down during drawing of the stope, the miners stand on the top of broken ore while placing the timbers to support patches of insecure wall rock. After the ground has been secured thus, drawing and cleaning down are resumed.

Shrinkage stoping is applicable to bodies of strong, firm ore enclosed between firm walls that will not slab or slough off to any great extent after standing for a considerable time. The method is applied most frequently to relatively thin, tabular deposits dipping at angles greater than 50°, in which few waste inclusions occur and which have fairly regular walls. By carrying transverse stopes separated by pillars, it may also be applied to wide, tabular deposits as indicated above.

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[30] Dye, Robert E., Mining Practice and Costs of the Vipond Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, p. 7 and fig. 6.

The "forced-caving system" may be considered a variation of this

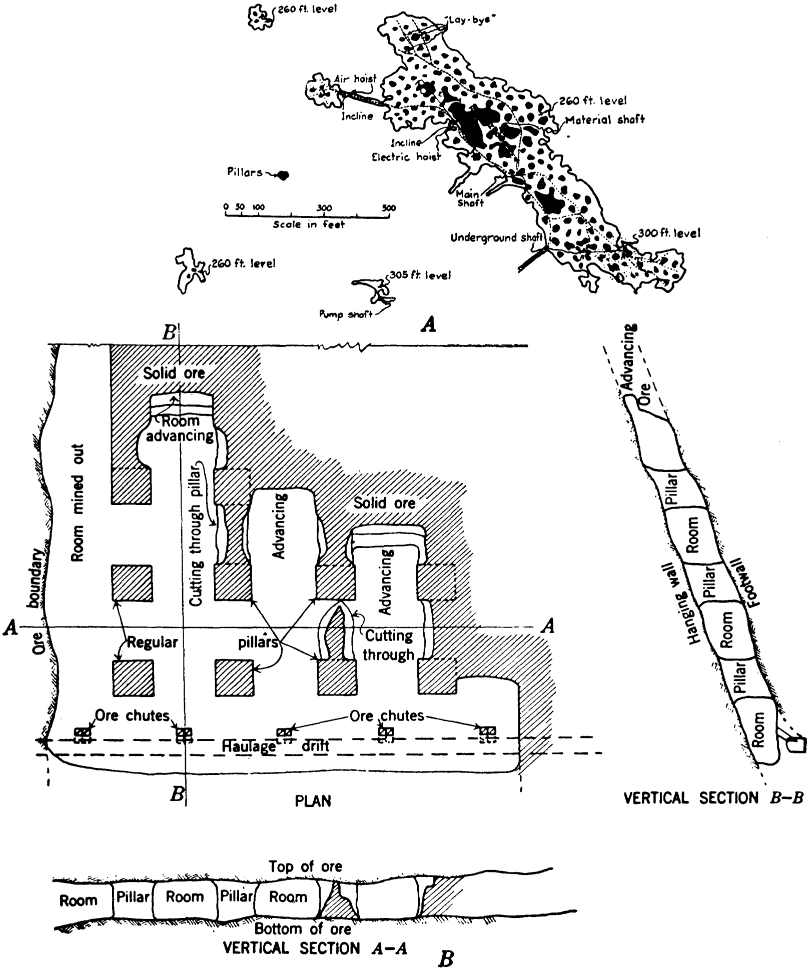
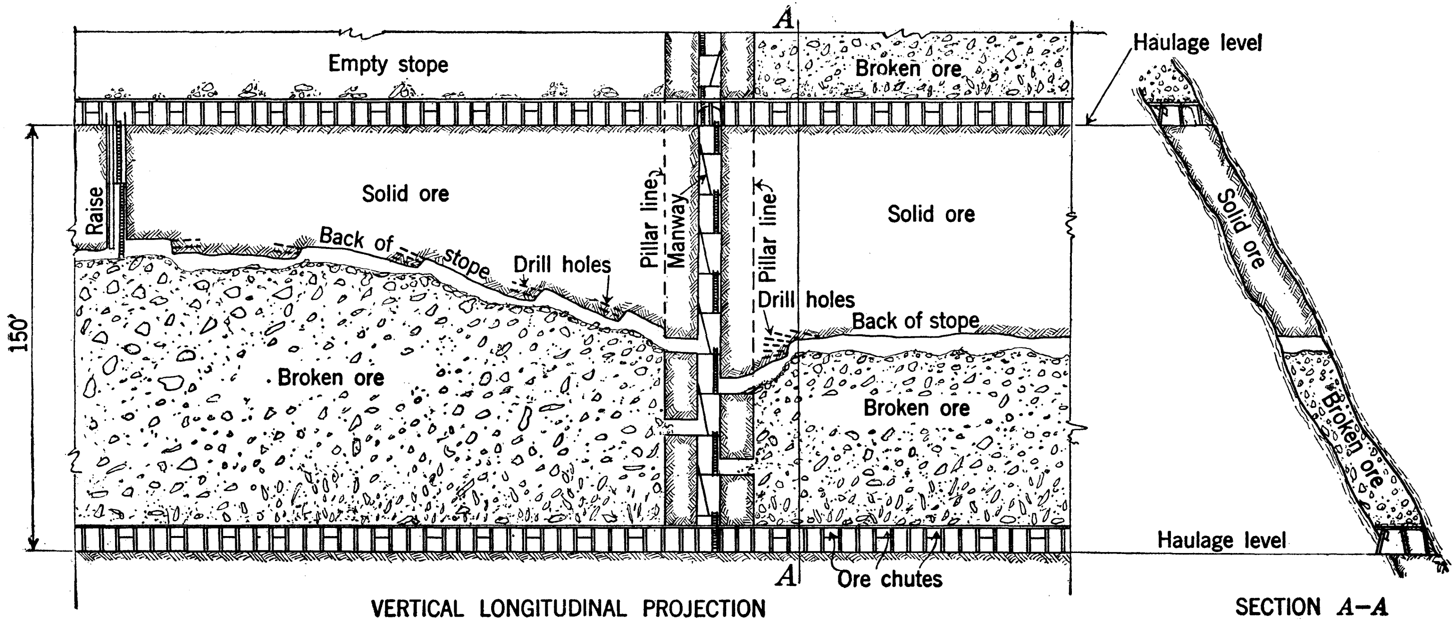

260 ft. level
"Lay-bys"
Air hoist
260 ft. level
Incline
Material shaft
Incline
Electric hoist
Pillars
0 50 100 300 500
Scale in feet
Main Shaft
Underground shaft
300 ft. level
260 ft. level
305 ft. level
Pump shaft
A
B
B
Solid ore
Room advancing
Room mined out
Cutting through pillar
Advancing
Solid ore
Advancing
Ore boundary
A
A
Regular
pillars
Cutting through
Ore chutes
Ore chutes
Haulage drift
PLAN
Advancing
Ore
Pillar
Room
Hanging wall
Pillar
Footwall
Room
Pillar
Room
VERTICAL SECTION B–B
Top of ore
Room
Pillar
Room
Pillar
Room
Bottom of ore
VERTICAL SECTION A–A

FIGURE 87.—*A*, Open stoping with casual pillars. *B*, Open stoping with regular pillars.

method and is employed for the nonselective mining of large, thick masses of ore at the Alaska-Juneau, Beatson, Climax Molybdenum, Britannia, and other mines. Figure 89 depicts the mining cycle used at Alaska-Juneau and figure 90 that at Beatson. In both cases the ore is broken down by large blasts into stopes that are kept partly full of broken ore (as in shrinkage stoping). The large blasts break ore directly into the stopes and have the further effect of shattering additional ore, part of which then caves. At both mines the miners


VERTICAL LONGITUDINAL PROJECTION

SECTION *A–A*

A
Empty stope
Broken ore
Haulage level
Raise
Solid ore
Back of stope
Drill holes
Pillar line
Manway
Pillar line
Drill holes
Back of stope
Solid ore
Broken ore
Broken ore
150′
A
Ore chutes
Solid ore
Broken ore
Haulage level

FIGURE 88.—Example of shrinkage stoping on drift sets.

come down through raises from the level above. At Juneau, small powder drifts (sometimes termed "coyote" drifts) are driven from the raises. Charges averaging 4,000 pounds of 40-percent dynamite are placed in the powder drifts segregated in two piles 35 or 40 feet apart. Three to five such piles make up the usual blast. At Beatson, however, a similar result is accomplished by drilling, from enlargements in the raises, a large number of long holes, which are loaded and fired in one blast.

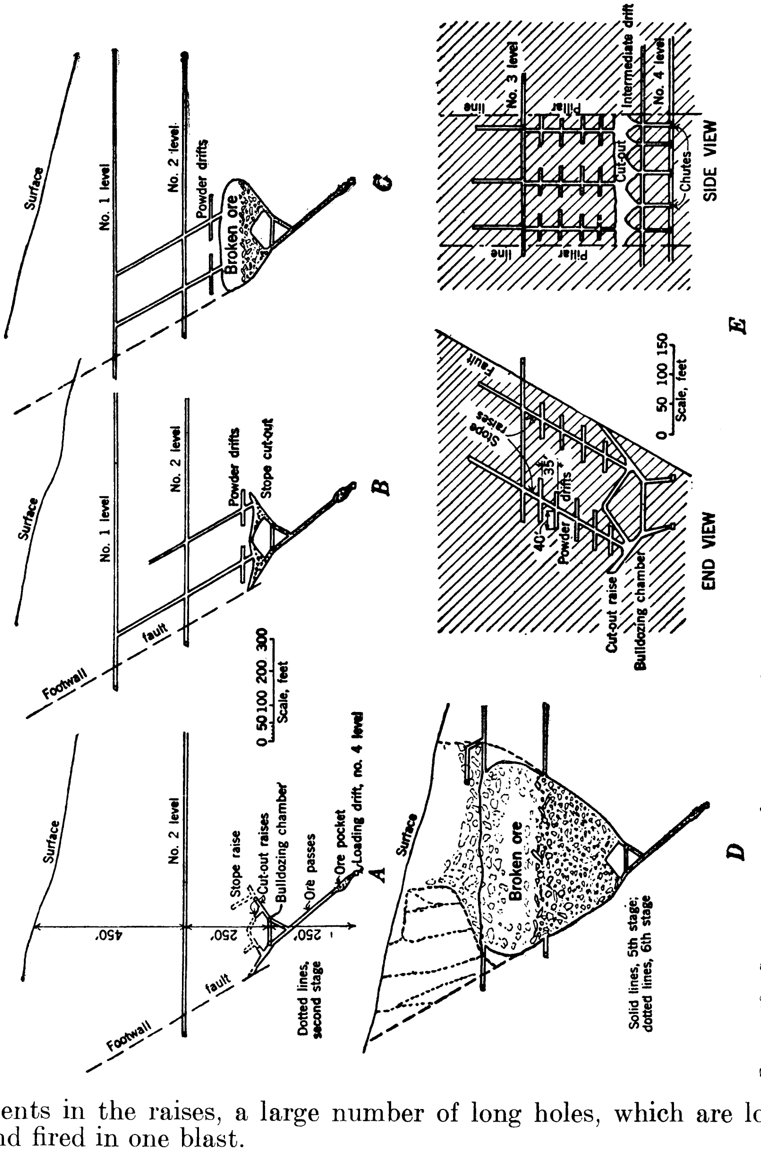

Surface
No. 1 level
No. 2 level
Powder drifts
Broken ore
C
Surface
No. 1 level
No. 2 level
Powder drifts
Stope cut-out
B
fault
Footwall
0 50 100 200 300
Scale, feet
Surface
No. 2 level
Stope raise
Cut-out raises
Bulldozing chamber
Ore passes
Ore pocket
Loading drift, no. 4 level
A
450'
250'
250'
fault
Footwall
Dotted lines, second stage
Surface
Broken ore
D
Solid lines, 5th stage; dotted lines, 6th stage
Fault
Stope raises
35'
40'
Powder drifts
Cut-out raise
Bulldozing chamber
0 50 100 150
Scale, feet
END VIEW
E
No. 3 level
Intermediate drift
No. 4 level
Pillar line
Pillar line
Cut-out
Chutes
SIDE VIEW

FIGURE 89.—Sequence of stope development and stoping operations, Alaska-Juneau mine, Alaska. *A* First and second stages. *B*, Third stage. *C*, Fourth stage. *D*, Fifth and sixth stages. *E*, Fully developed stope, north ore body, ready for powder-drift blasting.

For the successful use of the ordinary shrinkage-stoping method the ore must be strong and stand well, as the back of the stope under which the men must work is unsupported except for occasional props or stulls that may be set to hold local patches of loose ground. Although the broken ore left in the stope affords support to the walls until drawing begins, unless the walls are firm they may slough or even collapse during and after drawing and cause serious dilution of the ore.

2. CUT-AND-FILL STOPING

In cut-and-fill stoping the ore is excavated by successive flat or inclined cuts or slices, working upward from the level as in shrinkage stoping; but after each cut, all the broken ore is removed and waste rock, sand, or other filling material is run in and the excavation filled to within a few feet of the back, thus providing permanent support to the walls (fig. 91) and a working floor for the next cut. The term "cut-and-fill" implies a definite and characteristic sequence of oper-

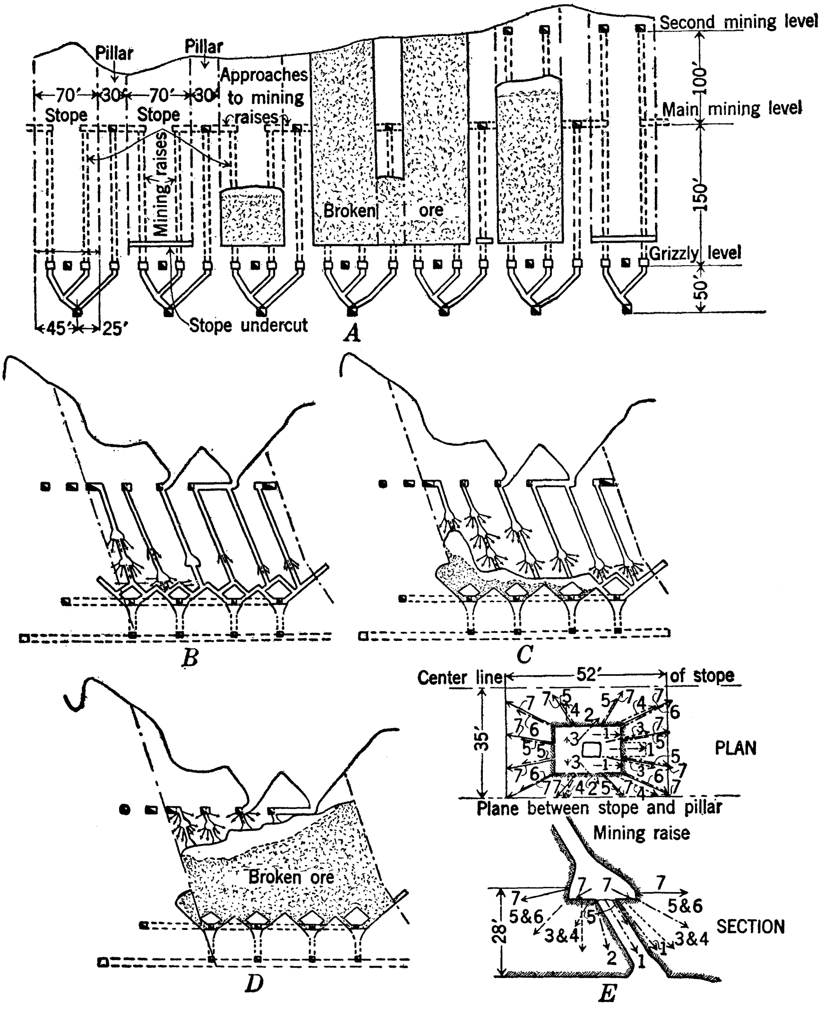
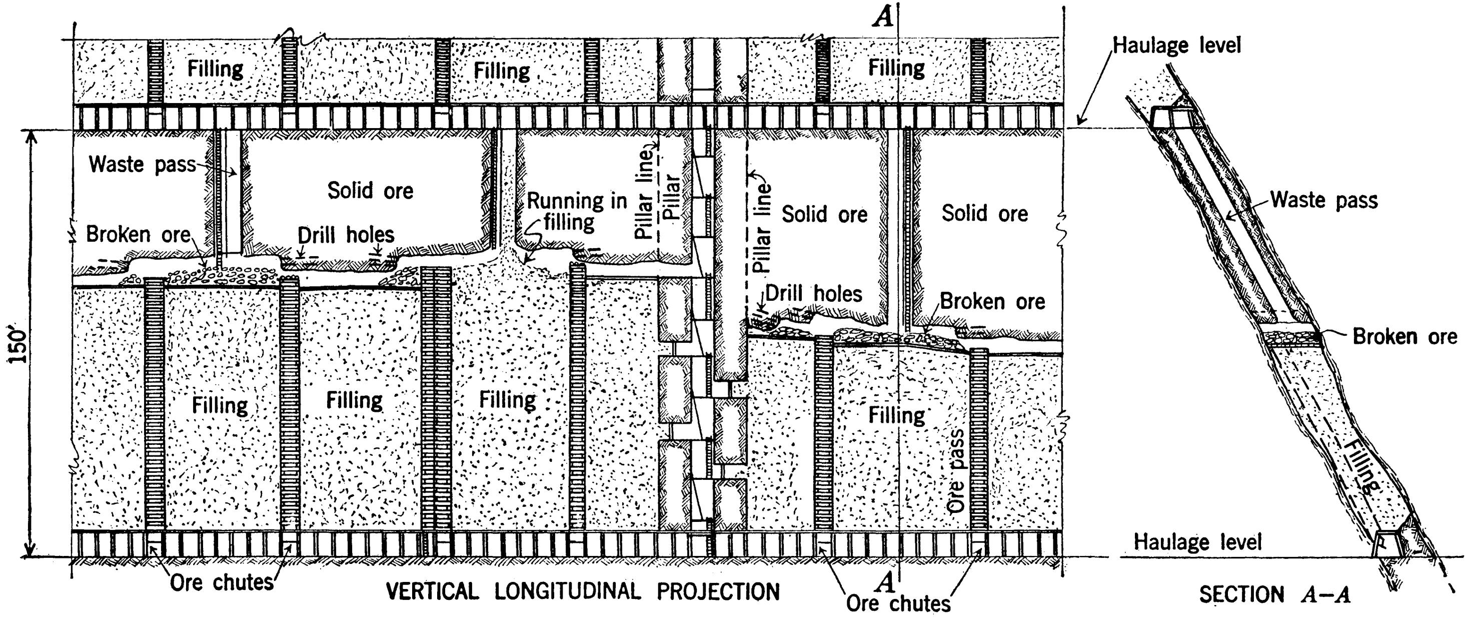

Second mining level
Pillar
Pillar
Approaches to mining raises
70'
30'
70'
30'
Stope
Stope
Mining raises
Main mining level
100'
150'
Broken ore
Grizzly level
50'
45'
25'
Stope undercut
A
B
C
Center line
52'
of stope
35'
PLAN
Plane between stope and pillar
Mining raise
28'
SECTION
7
5&6
3&4
2
1
D
E

FIGURE 90.—Mining system employed at Beatson mine, Latouche, Alaska. *A*, Ideal vertical section parallel to strike showing levels, branch raises, stopes, and pillars. *B*, Section of 202 stope, preparation completed and stope undercut. *C*, Section of 202 stope after 2 months' mining. *D*, Same stope after 6 months' mining. *E*, Plan and section of typical long round in raise bench in hard ground.


A
Filling
Filling
Filling
Haulage level
Waste pass
Solid ore
Running in filling
Pillar line
Pillar
Pillar line
Solid ore
Solid ore
Broken ore
Drill holes
Drill holes
Broken ore
150'
Filling
Filling
Filling
Filling
Ore pass
Ore chutes
Ore chutes
A
Waste pass
Broken ore
Filling
Haulage level

VERTICAL LONGITUDINAL PROJECTION

SECTION A–A

FIGURE 91.—Example of horizontal cut-and-fill stoping.

ations: (1) Breaking a slice of ore from the stope back, (2) removing the broken ore, (3) introducing filling; then (1) breaking again, and so on. The filling is employed primarily to support the walls of the stope and may consist of waste sorted from the ore in the stope, waste blasted from the walls of the lode adjacent to the stope, waste rock from development work or special waste stopes, rock from surface quarries or glory holes, sand and gravel, mill tailings, or slag. Stulls, props, and cribs may be erected on top of the fill to support local patches of loose or insecure ground.

Cut-and-fill stoping is applicable to the mining of firm ore enclosed within walls, one or both of which are weak and heavy. The deposits may be tabular in form, dipping at angles steeper than the angle of repose of the broken ore (or flatter, if scrapers are employed for moving the ore in the ore passes), or to wide, thick ore bodies or masses. Generally speaking, it is suitable for mining deposits too irregular for shrinkage stoping and deposits in which shrinkage could be employed were it not for the fact that the walls are too weak.

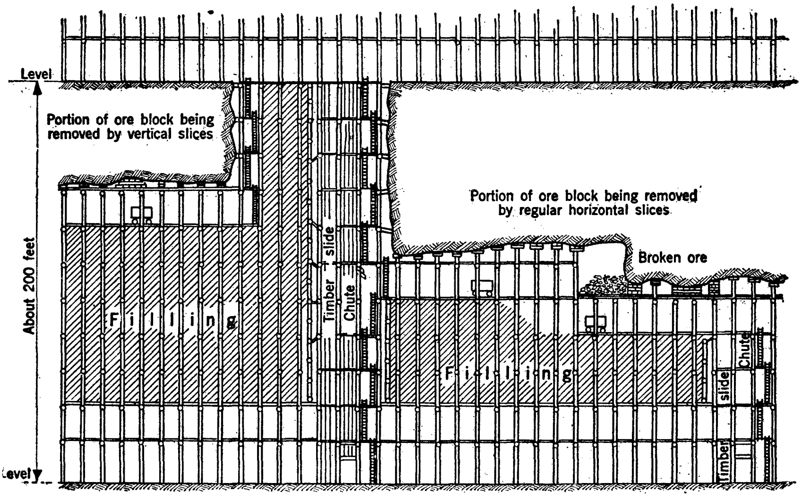

Level
Portion of ore block being removed by vertical slices
Portion of ore block being removed by regular horizontal slices
Broken ore
About 200 feet
Filling
Timber slide
Chute
Filling
Chute
slide
Timber
Level

FIGURE 92.—Stringer-set cut-and-fill stoping.

The method is also used for mining high-grade veins that are less than minimum practicable stoping width and therefore require the breaking of wall rock to provide working room. When the ore and walls are broken separately, the method is termed "resuing" or "stripping." In resuing, one wall may be drilled and shot down first and the rock left in the stope for fill, after which the vein is "stripped" from the side of the resulting excavation; or the vein may be shot down first, the broken ore removed, and the wall then stripped to provide working room and filling for the stope.

Since the filling material does not directly support the stope back, but only the walls, the ore itself must stand well for a short time, at least so as to apply this method without the use of timber for auxiliary support. In veins of moderate width a stringer-set cut-and-fill variation (fig. 92) of the general method is employed sometimes where the walls are heavy; the ore itself is fairly strong, but the ground is not so heavy as to require the use of square-set-and-fill stoping.

In fairly wide ore the back may tend to slab off, and temporary stringers supported by posts resting on the fill may be employed to afford back and wall support above the filling (fig. 93).

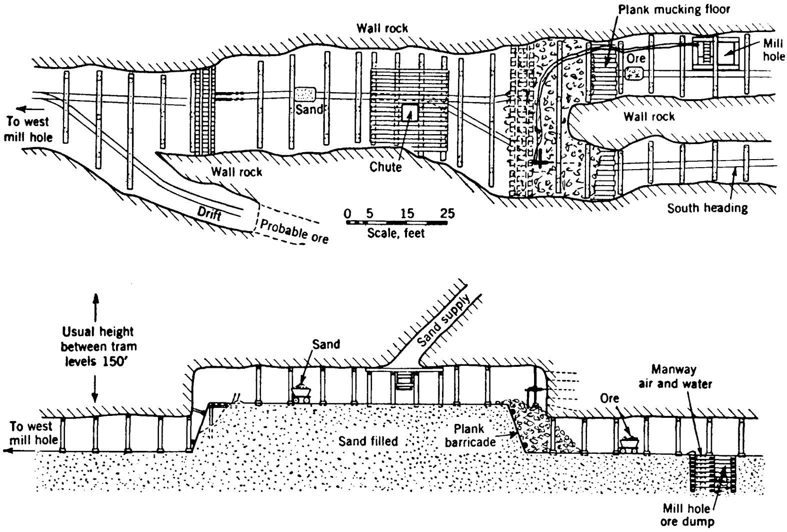

Plank mucking floor
Wall rock
Mill hole
Ore
To west mill hole
Sand
Wall rock
Wall rock
Chute
Drift
Probable ore
0 5 15 25
Scale, feet
South heading
Usual height between tram levels 150'
Sand supply
Sand
Manway air and water
Ore
To west mill hole
Sand filled
Plank barricade
Mill hole ore dump

FIGURE 93.—Horizontal cut-and-fill stoping with temporary timber support for back.

If both the walls and the ore itself are heavy and weak, square-set stoping or a caved-stope method would be indicated rather than cut-and-fill stoping.

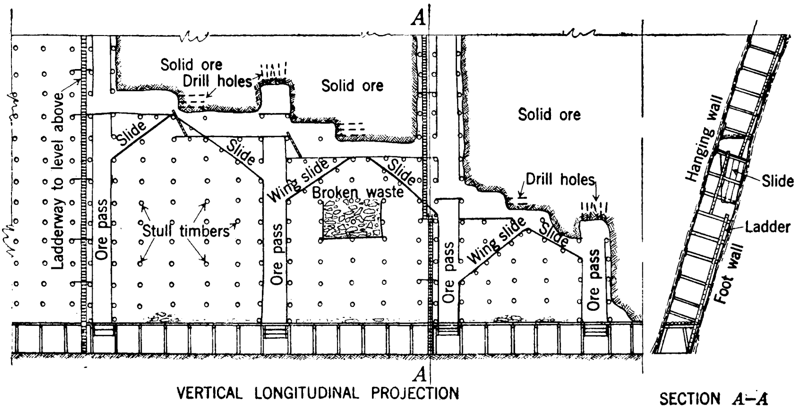

A
Solid ore
Drill holes
Solid ore
Solid ore
Slide
Slide
Wing slide
Slide
Broken waste
Drill holes
Wing slide
Slide
Ladderway to level above
Ore pass
Stull timbers
Ore pass
Ore pass
Ore pass
Hanging wall
Slide
Ladder
Foot wall
A
VERTICAL LONGITUDINAL PROJECTION
SECTION A–A

FIGURE 94.—Example of stull stoping in narrow vein.

3. STULLED STOPES IN NARROW VEINS

Casual stulls may be used locally in open stopes or in other forms of supported stopes, as already indicated, and in such instances are only incidental to the mining method. In some narrow or moderately wide lodes, however, a regular arrangement of stull timbers is employed to support the immediate hanging wall and constitutes a rather distinct method of mining. It is especially applicable where the ore is separated from the solid or true wall (or walls) by one to a few feet of barren rock that loosens and slabs off after being exposed for a short time unless held in place by stulls—in other words, where only nominal rather than great weight develops. Where a weak hanging wall extends back a considerable distance from the vein, the weight usually will be too great to be held in this manner and cut-and-fill or square-set stoping must be employed. Figure 94 shows a stull-supported stope.

4. SQUARE-SET STOPING

In square-set stopes the walls and back of the excavation are supported by regular framed timbers forming a skeleton enclosing a series of contiguous, hollow, rectangular prisms in the space previously occupied by the ore and providing continuous lines of support in three directions at right angles to each other (fig. 95).

The ore is excavated in small rectangular blocks just large enough to provide room for standing a set of timber. The essential timbers comprising a standard square set are termed, respectively, "posts," "caps," and "girts" (or "ties"). The posts are the upright members and caps and girts are horizontal members. The ends of each member are framed to give each a bearing against the other two members at the corners where they meet.

Ordinarily the stopes are mined in floors or horizontal panels, one above the other, and the sets of each floor are framed into the sets of the preceding floor (fig. 95, at *B*). Sometimes, however, the sets are mined in a series of vertical or inclined panels. Square-set stoping usually is accompanied by filling, and often in heavy ground the sets are filled with waste promptly after they are installed, leaving only a small volume of unfilled stope at any time. It has come to be accepted quite generally that unless the ground is heavy enough to require filling for permanent support, the expense of square-setting is not warranted and some other method should be employed.

Square-setting is adapted to mining regular or irregular ore bodies, commonly on dips steeper than about 45°, where the ore and/or walls are too weak to stand even over short spans for more than a brief time, and where caving and subsidence of overlying rocks must be prevented. From the viewpoint of support alone, both temporary and permanent, the method is applicable to conditions no other method has yet been devised to meet. Where the overlying strata may be allowed to cave, and where the loss of some ore or some dilution with waste would not be a serious drawback, caved-stope methods are adaptable to as bad or worse ground conditions and cost less.

Square-set stoping often is used in combination with other methods for extracting pillars between or over filled stopes. It is also employed as an auxiliary method, as for supporting the sill and first floors of some cut-and-fill or shrinkage stopes.

The stopes usually are worked upward from the level either in horizontal or inclined floors or in successive vertical panels, one alongside the other, but in some instances, especially in loose, running ground, may be worked from the top downward (underhand square-setting, fig. 96).

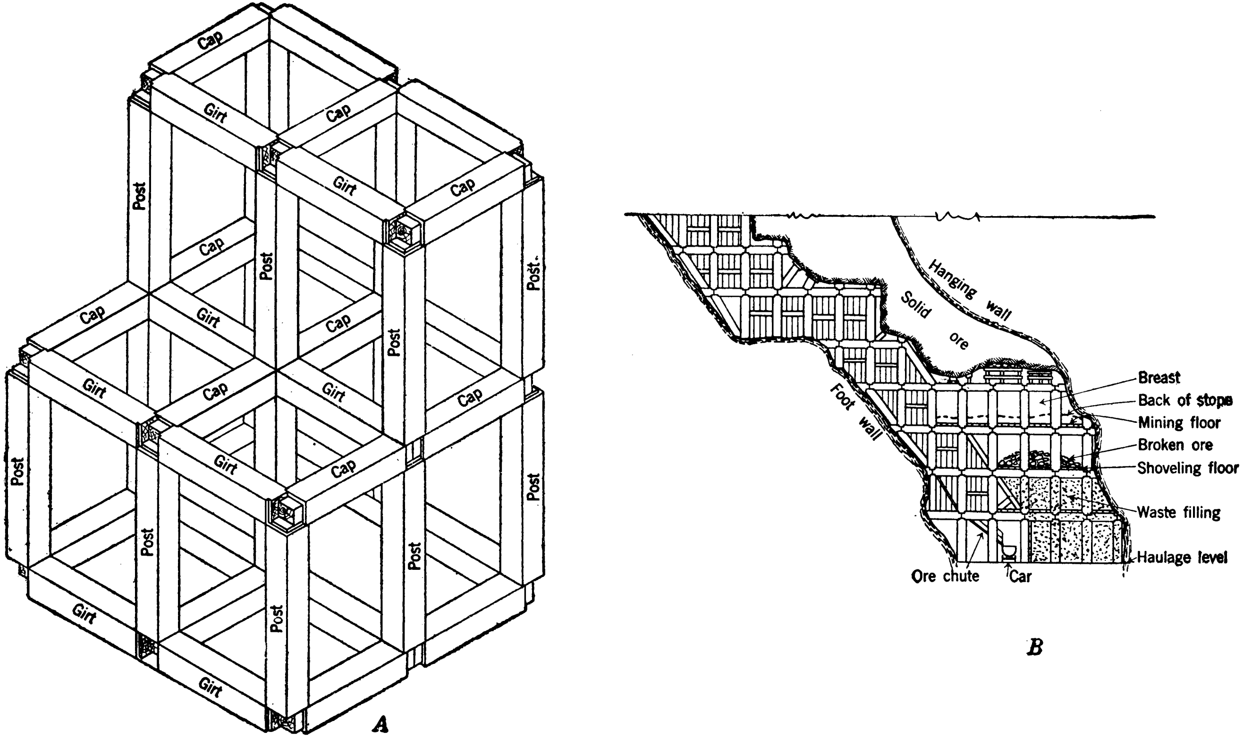

Cap
Girt
Post
A
Hanging wall
Solid ore
Foot wall
Breast
Back of stope
Mining floor
Broken ore
Shoveling floor
Waste filling
Haulage level
Ore chute
Car
B

FIGURE 95.—Square-set stoping. *A*, Square-set timbering. *B*, Vertical transverse section through typical square-set stope.

C. CAVED STOPES

When caved-stope methods are employed, breaks to and subsidence of the surface will occur ultimately if caving is continued over an area wide and thick enough in relation to the depth of cover. Hence, caved-stope methods are applicable only where there is no objection to caving the overlying strata or to surface subsidence.

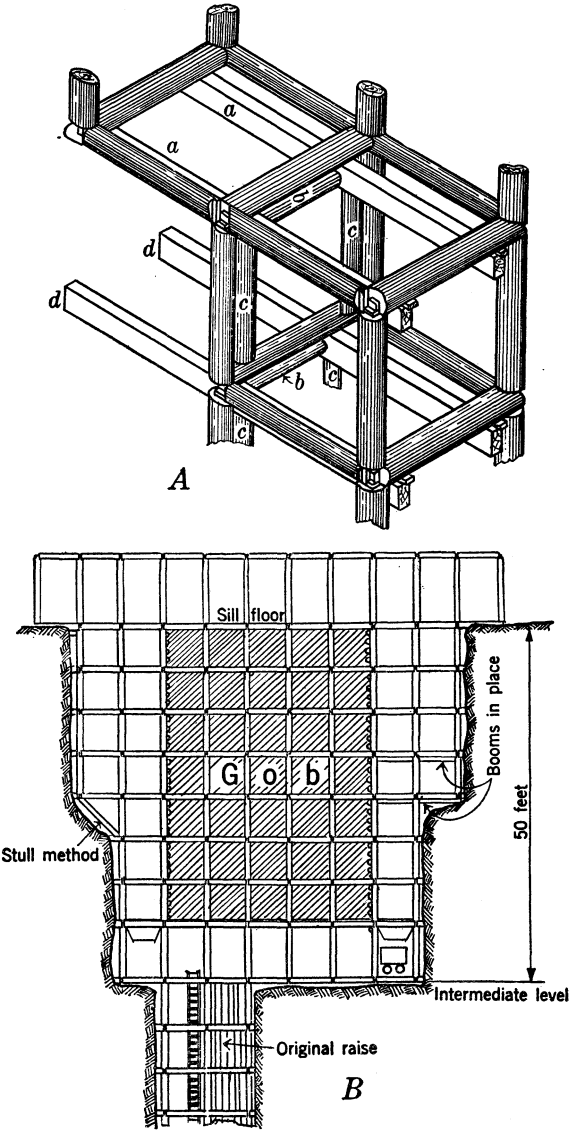

a
a
b
c
d
d
c
c
b
c
c
A
Sill floor
Gob
Booms in place
50 feet
Stull method
Intermediate level
Original raise
B

FIGURE 96.—Underhand square-set stoping, Black Rock mine, Mont. *A*, Method of using booms. *B*, General section through stope.

Caved stopes are of two distinct types: In the first, the ore is broken by caving induced by undercutting a block of the ore and isolating it from or weakening its connection to the surrounding ore or walls; in the second, the ore itself is broken by conventional drilling and blasting methods in a series of horizontal or inclined slices, and the capping is allowed to cave into and fill the space occupied previously by the ore. The first type embraces the caving methods of mining, whereas the second comprises the top-slicing method.

1. BLOCK CAVING

In block caving (fig. 97) a thick block of ore is partly cut off from surrounding blocks or the walls of the ore body by a series of drifts, one above the other, or by boundary shrinkage stopes; it is then undercut by removing a slice of ore or a series of slices separated by small pillars underneath the block. The isolated, unsupported block of ore breaks and caves under its own weight. The ore so broken is drawn off from below; and, as the caved mass moves downward owing to continued drawing from below, it is broken further by pressure and attrition. The capping finally caves also and follows the column of broken ore downward.

In the earliest applications of this method the block was undercut on or immediately above the haulage level, and the caved ore was shoveled into cars in drifts driven under or spiled through the cave. This system entailed the driving and maintenance of a large number of drifts to recover the ore and has been superseded by caving to a system of chutes or branched raises extending from haulage drifts at some distance below the bottom of the undercut and with an intermediate grizzly level on which chunks of ore too large to pass through the grizzlies are broken by sledging and bulldozing. With this system, hand shoveling is virtually eliminated.

Natural forces are utilized to the highest degree possible for breaking the ore, transferring the broken ore to the haulage levels, and filling the mined-out areas.

The method is a nonselective one, and lean sections of the ore body and even barren horses of waste are broken up and drawn with the ore. It is applicable to the wholesale mining of masses of ore large in three dimensions that will cave readily when support is removed by undercutting, and which, after caving, will break up fine enough to pass through the extraction raises as the caved mass is drawn downward. This condition occurs when the ore is friable enough to break up readily or the ore body is traversed by a multiplicity of closely spaced seams, fracture and joint planes, or invisible planes of weakness that strike and dip in various directions.

Block caving is employed principally for mining low-grade ores where the inevitable loss of some ore and some dilution with capping and wall rock are permissible and where mining must be done on a large scale and at a lower cost than can be obtained by the use of supported-stope or other selective mining methods.

2. SUBLEVEL CAVING

In sublevel caving, relatively thin blocks of ore are caused to cave by successively undermining small panels. The ore body is developed by a series of sublevels usually spaced at vertical intervals of 18 to 25 or 30 feet (more recently, intervals of 50 feet or more have been

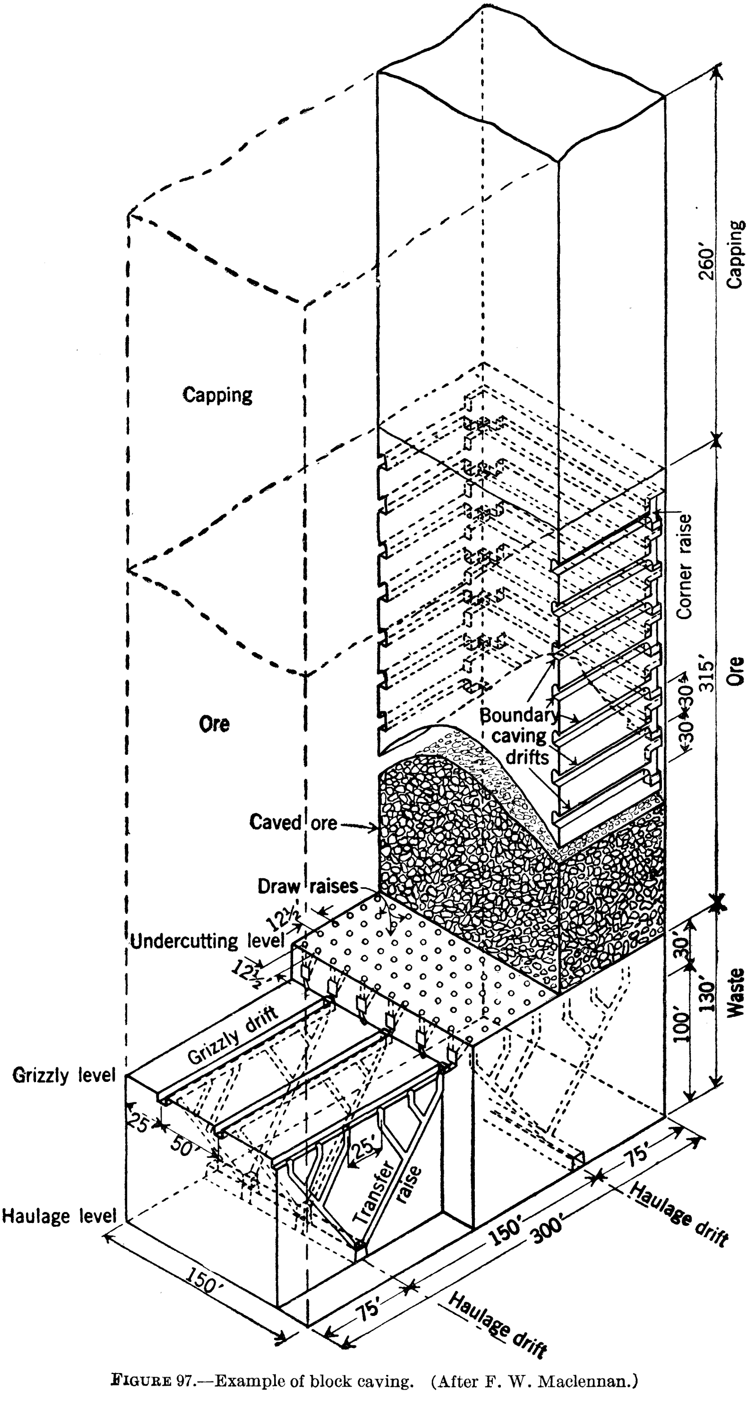

Capping
Ore
260'
Capping
Corner raise
315'
Ore
30'
30'
30'
Boundary caving drifts
Caved ore
Draw raises
12½'
Undercutting level
12½'
Grizzly drift
Grizzly level
25'
50'
25'
Transfer raise
Haulage level
150'
75'
Haulage drift
150'
300'
75'
Haulage drift
30'
130'
100'
Waste

FIGURE 97.—Example of block caving. (After F. W. Maclennan.)

used). Figure 98 (*A*, *B*, and *C*) shows complete development of a block or panel. Customarily all of this work is not done before stoping begins, but usually only one or, at most, two sublevels are opened at a time, beginning at the top of the ore. Sublevels are developed by first connecting the tops of the raises with a longitudinal drift from which timbered slice drifts are driven right and left to the ore boundaries or to the limits of the panel. Usually, alternate drifts are driven first, and caving back from them is begun and continued toward the raises while the intermediate slices are being driven. The bottoms of the slices or crossdrifts are covered with poles or plank, laid on cross timbers to form a good floor over which to scrape and to hold back the gob when caving the next sublevel below.

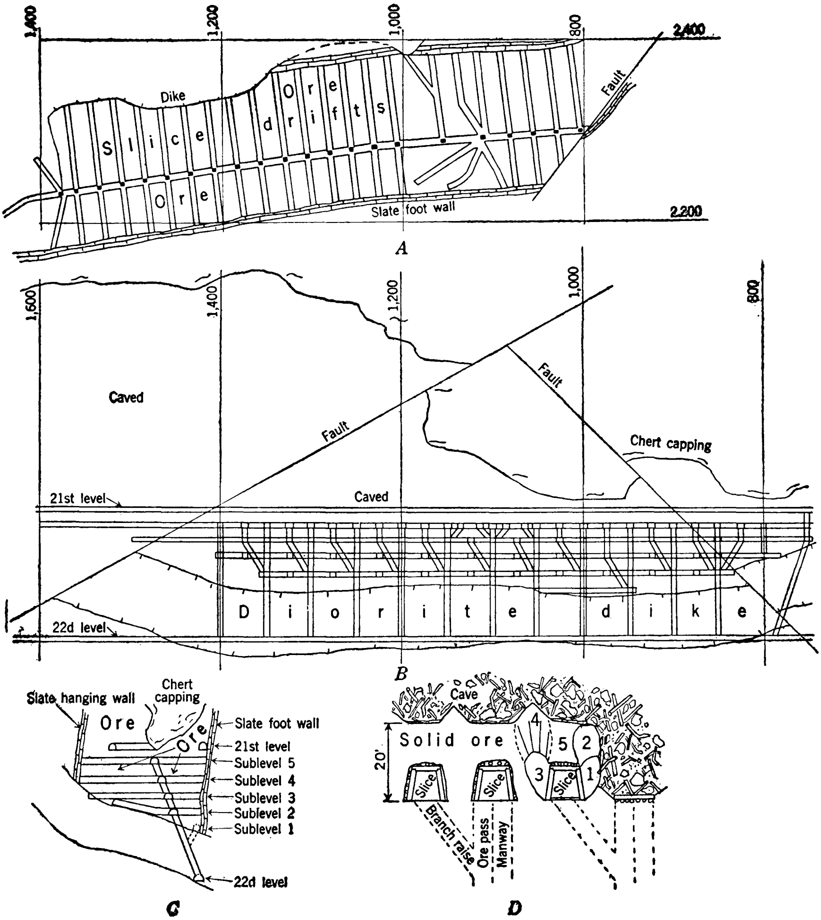

1,400
1,200
1,000
800
2,400
Dike
Slice
Ore drifts
Fault
Ore
Slate foot wall
2,200
A
1,600
1,400
1,200
1,000
800
Caved
Fault
Fault
Chert capping
21st level
Caved
Diorite dike
22d level
B
Chert capping
Slate hanging wall
Ore
Ore
Slate foot wall
21st level
Sublevel 5
Sublevel 4
Sublevel 3
Sublevel 2
Sublevel 1
22d level
C
Cave
Solid ore
20'
4
5
2
3
1
Slice
Slice
Slice
Branch raise
Ore pass
Manway
D

FIGURE 98.—Example of sublevel caving. *A*, *B*, and *C*. Development of ore body for caving. *D*. Cross section showing details of caving back, cuts blasted in order shown.

Caving is begun at the ends of the slices by blasting cuts, as shown in figure 98, *D*, and is continued in successive stages, working out toward the raises. The broken ore is dragged by power scrapers into the raises.

More recent practice in at least one mine is to cave back two adjacent slices at a time, using a 50-foot instead of a 25-foot sublevel interval and an entrance manway to the cave, as shown in figure 99.[31] When the top sublevel or a section of it has been mined out work is begun on the next sublevel below.

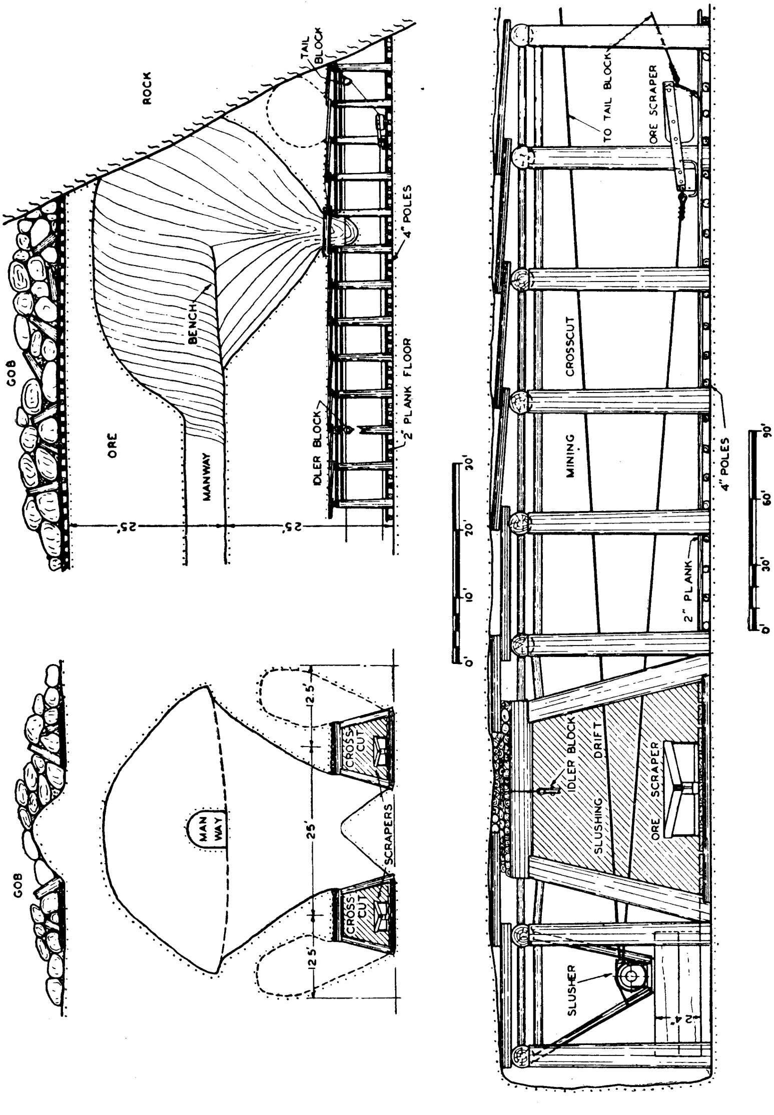

ROCK
TAIL BLOCK
4" POLES
2" PLANK FLOOR
IDLER BLOCK
BENCH
ORE
MANWAY
GOB
25'
25'
GOB
MAN WAY
CROSS CUT
SCRAPERS
CROSS CUT
12.5'
25'
12.5'
0' 10' 20' 30'
TO TAIL BLOCK
ORE SCRAPER
CROSSCUT
MINING
4" POLES
2" PLANK
IDLER BLOCK
SLUSHING DRIFT
ORE SCRAPER
SLUSHER
4'
0' 30' 60' 90'

FIGURE 99.—Sublevel caving, large-pillar method.

Sublevel caving may be employed for mining smaller deposits than would be suited to block caving, but is also applicable to large ore bodies and to soft ores, which will stand fairly well for a brief time over short spans but will cave over wider openings. The capping

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[31] Bowen, R. A., Sublevel Caving, Large-Pillar Method, at the Montreal Mine: Mining Technology, Am. Inst. Min. and Met. Eng., Tech. Pub. 886, March 1938.

(and later the gob) must be of such character that it will hang up long enough to permit removal of the caved ore from beneath it in safety and without serious attendant dilution with waste. Schaus[32] has summarized the conditions necessary for successful sublevel caving in the iron mines of the Gogebic range, as follows:

> The determining conditions to which this method is adapted are (*a*) a dipping and pitching irregular ore body, which does not lend itself to top slicing; (*b*) a medium-soft ore which breaks fine yet stands well and is not free-caving; (*c*) a hard capping which caves in medium-sized blocks without much fines and which is easily controlled.

The method is also employed successfully in hard ores and may be used under conditions where top slicing would be dangerous because of the hanging up of the gob. For successful block caving the ore should be free-running; that is, it must be of such a nature that it will not pack and form a semisolid mass that cannot be drawn off through the draw raises. Due to the relative thinness of the slices and the small size of the units caved in sublevel caving, packing is not so serious an obstacle and the method can be used for mining softer and somewhat stickier ores than can block caving.

3. TOP SLICING

The term "top slicing" is applied to the method of mining whereby the ore is extracted by excavating a series of horizontal (sometimes inclined) timbered slices alongside each other, beginning at the top of the ore body and working progressively downward; the slices are caved by blasting out the supporting timbers or allowing them to crush, bringing the capping or overburden down upon the bottoms of the slices, which have previously been covered with a floor or timber mat to separate the cave from the solid ore beneath and prevent admixture of waste with the ore (figs. 100 and 101). Successive slices are mined in a similar manner up to the overlying gob or mat, which consists of an accumulation of broken timbers, flooring, and lagging used in the overlying slices and of caved capping or overburden. As the slices are mined and caved, this mat follows the mining downward, filling the space formerly occupied by the ore. The mat also serves to control the movement of the caved overburden and to prevent dilution of the ore with barren capping.

Top slicing is applicable to the mining of soft, weak ore that will stand unsupported for only a very short time, even over narrow spans, and that is overlain by an unconsolidated overburden or a weak capping that will break up and cave promptly after removal of support, tightly filling the space formerly occupied by the slice and leaving no open holes. It may be applied in tabular deposits lying at all angles of dip, from flat to vertical, or in wide, thick ore bodies. It is used successfully in deposits of very irregular outline and is employed for recovering pillars of ore from between filled or caved stopes and for mining broken ore or old caves that could not be worked safely from the bottom upward. The method is similar in some respects to sublevel caving—the ore is mined from the top downward by a series of slices, and the overlying strata cave and follow the mining downward with a mat intervening between the caved overburden and the ore beneath. The methods differ, however, in that in top slicing the slice is mined right up to the floor of the caved slice above, whereas in sublevel caving the sublevel interval

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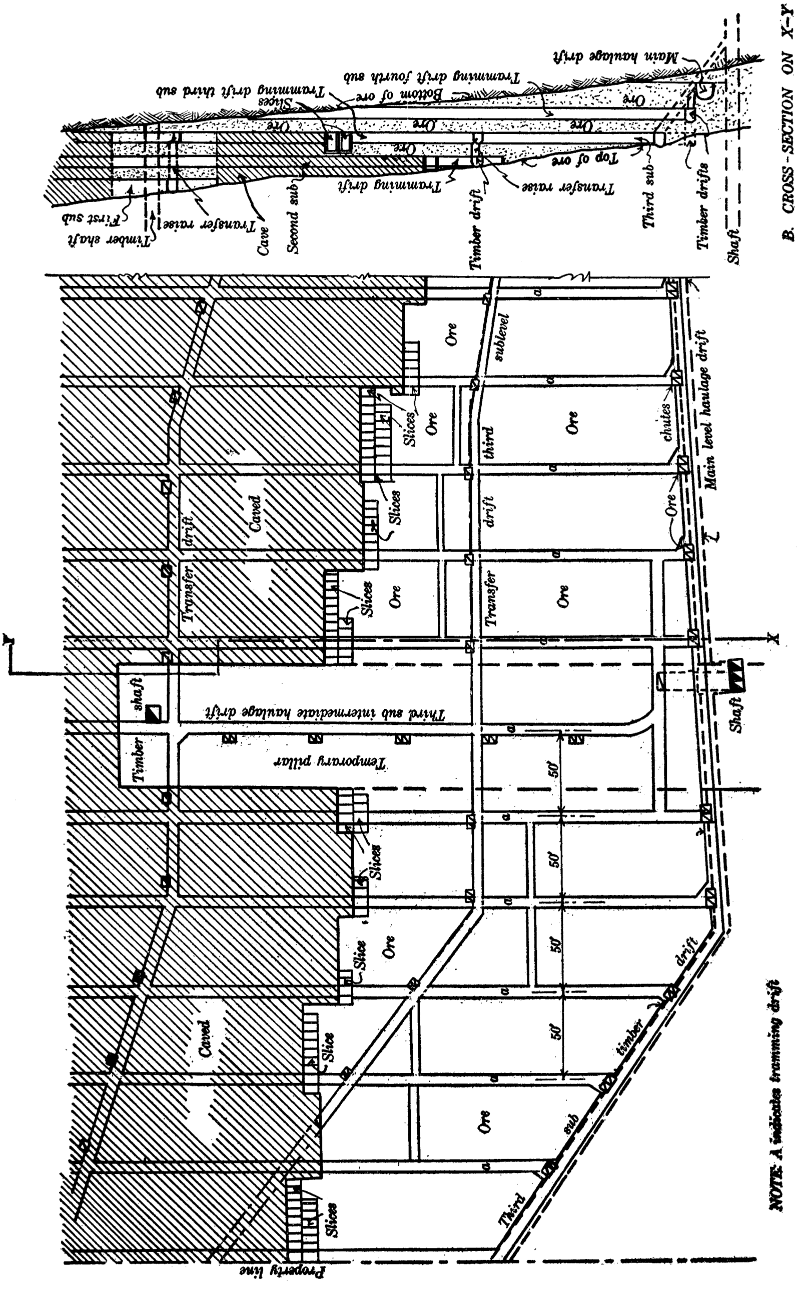
[32] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29. pp.


Main haulage drift
Tramming drift fourth sub
Bottom of ore
Slices
Tramming drift third sub
Ore
Top of ore
Transfer raise
Tramming drift
Timber shaft
First sub
Transfer raise
Cave
Second sub
Timber drift
Third sub
Timber drifts
Shaft
Ore
Third sublevel drift
Transfer drift
Caved
Slices
Slice
chutes
Main level haulage drift
Third sub intermediate haulage drift
Temporary pillar
Timber shaft
Shaft
50'
a
Third sub timber drift
Property line
X
Y
NOTE A indicates tramming drift

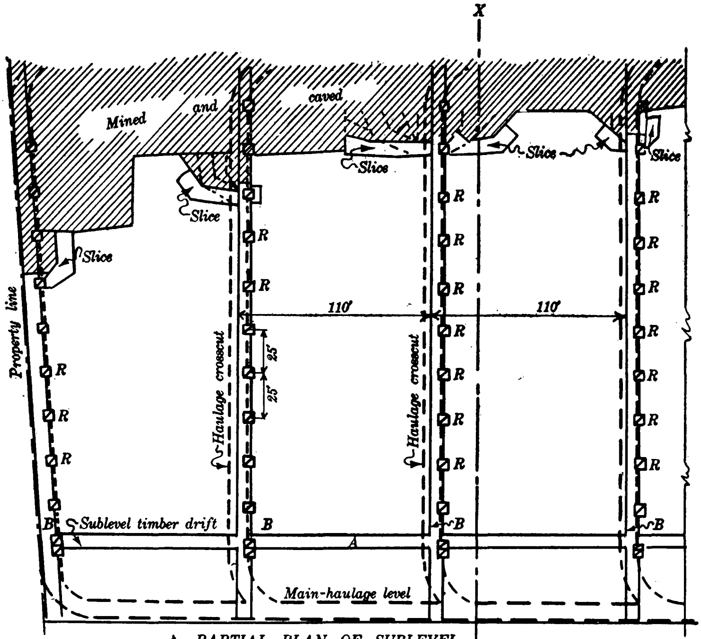
A. PLAN OF PART OF THIRD SUBLEVEL

B. CROSS-SECTION ON X-Y

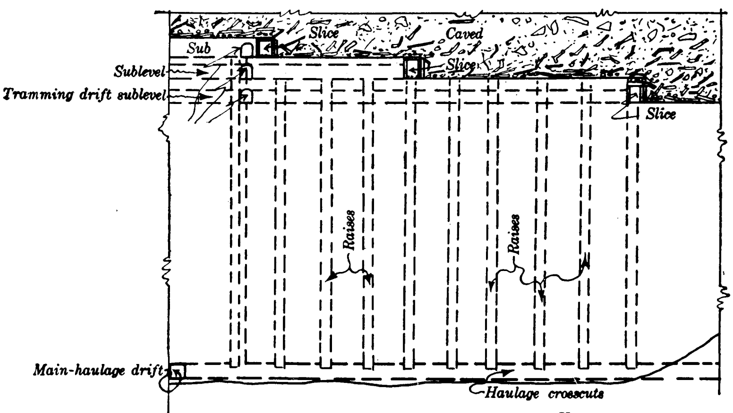
FIGURE 100.—Example of parallel top slicing, scraping to cars; flat-dipping ore body, 2 to 4 subs thick.

is greater (usually about twice), and in driving the slice drifts a back of ore is left over the tops of the drifts and below the mat of the slice above, which is extracted later by caving on the retreat. In sublevel caving the slice drifts are separated by intervening pillars, which are also mined on the retreat, whereas in top slicing they are driven immediately alongside each other.

Figure 100 shows the method of parallel top slicing in a relatively thin, flat-dipping ore body, and figure 101 shows radial top slicing to chutes in a thick ore body.


X
Mined and caved
Slice
Slice
Slice
Slice
Slice
R
Property line
Haulage crosscut
Haulage crosscut
110'
110'
25'
25'
Sublevel timber drift
B
B
B
B
A
Main-haulage level
Y

A. PARTIAL PLAN OF SUBLEVEL


Slice
Caved
Sub
Slice
Sublevel
Tramming drift sublevel
Slice
Raises
Raises
Main-haulage drift
Haulage crosscuts

B. CROSS-SECTION ON X–Y

FIGURE 101.—Radial top slicing to chutes; thick ore body.

The method has been applied most commonly in the mining of wide deposits of soft or weak ore overlain by a friable capping or unconsolidated overburden. A capping that breaks in large blocks that will wedge and arch over, leaving open holes in the gob, would be dangerous for top slicing. Hence, top-weight or vertical pressure is an essential condition for successful top slicing. The side walls may be either weak or strong.

Since top slicing requires the use of considerable timber and lagging, it is essential that there be a plentiful and cheap source of suitable timber available. Slice timber is required to stand for only a short time and is then blasted down or allowed to crush; a poorer grade is acceptable than would be required for more permanent supports, such as for drift timber. However, the timber should be tough and capable of standing considerable crushing before breaking.

A special application of the top-slicing method has been used for mining pillars between filled stopes under a capping that will not cave readily, even over large spans. In this instance, sand or waste filling is run in on top of the subsiding gob, keeping the hole filled. The filling will then act as a cushion to take the impact of any possible sudden collapse of the capping.

D. COMBINATIONS OF SUPPORTED AND CAVED STOPES

The use of two or more stoping methods independently of each other in the same mine, or even in different parts of the same ore body because of differences in the physical characteristics governing the requirements for support, does not constitute a combination method.

A combination method does exist, however, when a stoping system is laid out in advance of operations wherein two different methods are employed as essential complements, one to the other. An example is furnished where a series of alternating stopes and pillars is laid out, the stopes to be mined by shrinkage and then filled, or by cut-and-fill stoping, and the pillars to be mined by top-slicing between the filled stopes. Each method, one a supported-stope method and the other a caved-stope method, is essential to the complete extraction of the block of ore. Again, a block-caving system in which a considerable part of the ore is mined by shrinkage in undercutting stopes and boundary cut-offs and the balance by caving in reality constitutes a combination method. (It should be noted, however, that in modern block-caving practice only a small percentage, perhaps 2 to 5 percent of the ore, is customarily mined by shrinkage stoping.)

STOPING PRACTICE

The general conditions to which each of the principal stoping methods is applicable have been stated briefly in the preceding pages. Variations of the principal methods, differences in methods of drilling and blasting, handling ore and waste in the stopes, transferring broken ore from the stopes to the haulageways, and filling and timbering operations, and variations in sequence and direction of stoping, are employed to suit local conditions. The term "stoping practice" is employed in this bulletin to cover details of stoping procedure rather than the broader and more fundamental considerations that are covered by the term "stoping method."

VARIATIONS OF PRINCIPAL STOPING METHODS

Some of the variations of stoping methods have been discussed already. Thus, it has been noted that shrinkage, cut-and-fill, and square-set stopes may be mined by horizontal or inclined slices or cuts (angle of working); that square-set stopes usually are mined overhand, from the bottom upward, but may be mined underhand (direction of working); that in cut-and-fill stopes where the walls are very heavy a systematic stringer-set method of timbering may be used, or, if the backs are "scaly," temporary horizontal stringers on posts may be used (auxiliary support). Square-set stopes may be worked on successive horizontal floors or in vertical panels or sections (sequence and direction of working). In top-slicing, the slices may be driven parallel to each other or radially (figs. 100 and 101). It has also been noted that stopes may be carried longitudinally (along the vein) or, in wide ore, across the vein (direction of working).

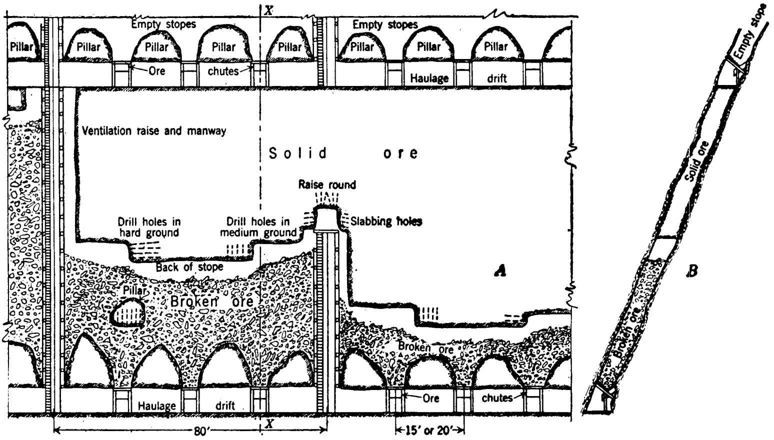

X
Empty stopes
Pillar
Pillar
Pillar
Pillar
Pillar
Empty stopes
Pillar
Pillar
Pillar
Ore
chutes
Haulage
drift
Ventilation raise and manway
Solid ore
Raise round
Drill holes in hard ground
Drill holes in medium ground
Slabbing holes
Back of stope
A
Pillar
Broken ore
Broken ore
Haulage
drift
Ore
chutes
80'
X
15' or 20'
Empty stope
Solid ore
B
Broken ore

FIGURE 102.—Shrinkage stoping on drift pillars, Nevada-Massachusetts mine, Nevada. *A*, Vertical, longitudinal projection. *B*, Vertical cross section, X=X.

Cut-and-fill or shrinkage stopes may be silled out full width on the level and the shrink or fill, as the case may be, carried on timbers, or silled a short distance above the level, thereby leaving a back or "arch" pillar over the drift to support the stopes. Modern practice usually favors the former method if the ore on the level is of good grade. This ore usually can be recovered more completely and at less cost before the stopes are worked out above it than by later robbing operations.

Any of the supported-stope methods may be employed for mining either on the advance or on the retreat (sequence of working). In advance mining, the stopes are started and carried upward or downward as development advances progressively farther and farther from the shaft or main adit. In retreat mining, the levels are developed to the extremities of the vein, deposit, or property boundary and stoping begins at the farthest points and progresses toward the shaft or adit. In advance mining, the haulageways become longer and longer and, if the ground is heavy, the cost of maintaining them under stoped ground becomes greater and greater. In retreat mining the active haulageways are in ground undisturbed by adjacent stoping, are therefore maintained more easily, and become progressively shorter. When mining is done on the retreat, the stoped-out areas can be abandoned and, unless movement and subsidence must be prevented, can be allowed to cave.

A few of the principal variations of stoping methods not heretofore described are those shown in figures 102 to 107, inclusive.

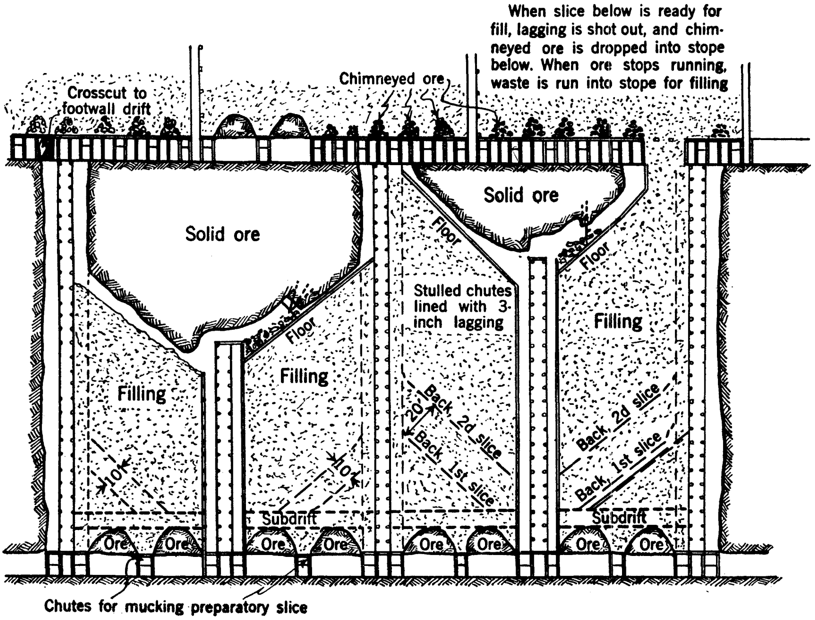

When slice below is ready for fill, lagging is shot out, and chimneyed ore is dropped into stope below. When ore stops running, waste is run into stope for filling
Chimneyed ore
Crosscut to footwall drift
Solid ore
Solid ore
Floor
Floor
Stulled chutes lined with 3-inch lagging
Filling
Filling
Filling
10'
10'
20'
Back 2d slice
Back 1st slice
Back 2d slice
Back 1st slice
Subdrift
Subdrift
Ore
Chutes for mucking preparatory slice

FIGURE 103.—Inclined cut-and-fill stoping, Eighty-Five mine, New Mexico.

Figure 102 shows shrinkage stoping on drift pillars at the Nevada-Massachusetts mine, Nevada. (Compare with fig. 88.)

Figure 103 depicts inclined cut-and-fill stoping at the Eighty-Five mine, New Mexico. (compare with figs. 91 and 93.) The inclined or "rill" variation of cut-and-fill stoping possesses the advantage of gravity movement of the ore to the ore passes and of filling into the stopes. It thus eliminates hand-shoveling as a major part of the operation and makes it unnecessary to lay track and use cars or wheelbarrows for handling the ore and for placing fill in the stopes. The rill system, however, does not permit much waste to be sorted out and left in the stopes; waste inclusions in the vein and slabs from the walls go into the chutes with the ore. Handling of ore and waste in horizontal stopes by means of power scrapers has increased during the past decade; and in large stopes, at least, it is doubtful whether gravity handling in rills is any cheaper than scraper handling.

Figure 104 illustrates the sublevel inclined cut-and-fill system developed at the Champion mine in Michigan. In changing from an advancing to a retreating system of mining, it was found that stoping had to be done on many levels to produce the necessary tonnage. By splitting the blocks between haulage levels with sublevels and mining as shown, concentrated stoping could be done and the output produced from one-third the number of levels otherwise required to be active. The figure shows the ore being removed by scrapers; where much sorting had to be done, however, it was necessary to lay track, load the ore into cars, and tram it to the ore chute to obtain best results.

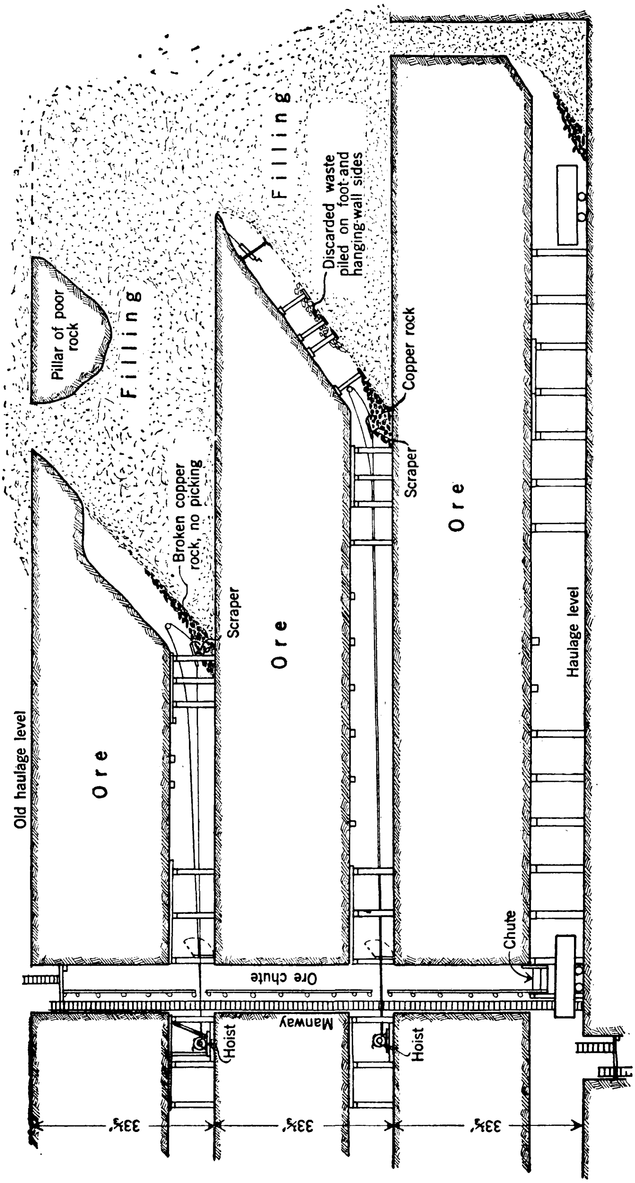

Old haulage level
Ore
Pillar of poor rock
Filling
Broken copper rock, no picking
Scraper
Ore
Filling
Discarded waste piled on foot and hanging-wall sides
Copper rock
Scraper
Ore
Haulage level
Chute
Ore chute
Manway
Hoist
Hoist
33½'
33½'
33½'

FIGURE 104.—Sublevel inclined cut-and-fill stoping, Champion mine, Michigan.

Figure 105 shows a semishrinkage stope that is a combination of shrinkage and cut-and-fill or shrinkage with delayed filling. Instead of cleaning out the ore and running in filling after each cut, two or more shrinkage cuts are made before the ore is drawn out and the stope filled. This system effects a large reduction in flooring expense.

The rilled square-set system is illustrated by the practice at the Bawdwin mine in Burma (fig. 106) described by Calhoun.[33] Inclining or rilling the stopes largely eliminates hand-shoveling, as both ore and fill run by gravity. The objection to rilling in cut-and-fill stopes, that is, the difficulty of sorting waste, does not apply to the same extent in square-set stoping, as sets can be kept open for the express purpose of providing a space to receive sorted waste.

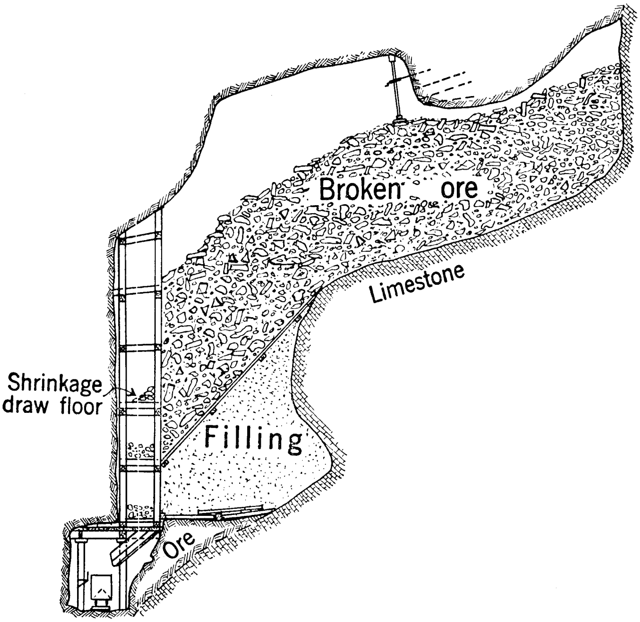

Broken ore
Limestone
Shrinkage draw floor
Filling
Ore

FIGURE 105.—Semishrinkage stope.

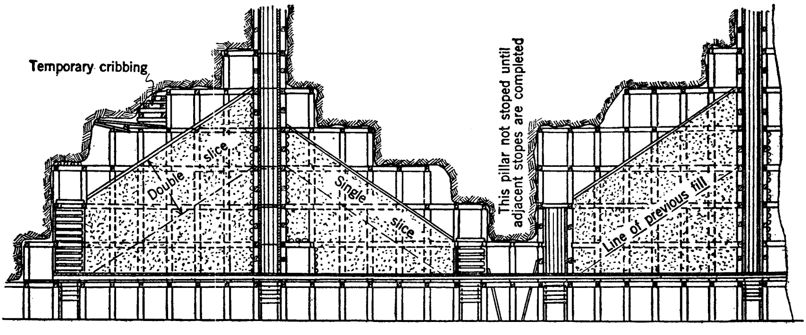

Temporary cribbing
Double slice
Single slice
This pillar not stoped until adjacent stopes are completed
Line of previous fill

FIGURE 106.—Rilled square-set system, Bawdwin mine, Burma.

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[33] Calhoun, A. B., Mining Methods at Bawdwin Mine: Trans. Am. Inst. Min. and Met. Eng. (issued with Min. and Met.), vol. 69, August 1923, pp. 208–247.

The square-set system employed at the Bunker Hill and Sullivan mine in Idaho is shown in figure 107. Stoping is begun on the hanging-wall side of the vein, as at *A*, and proceeds toward the footwall, carrying a nearly vertical stope face and a steeply arched back.

The foregoing and other variations are described and discussed in more detail in Bulletin 390, Stoping Methods and Costs, by Chas. F. Jackson and E. D. Gardner.

DRILLING AND BLASTING

Drilling in stopes is done with pneumatic-feed stopers, light hand-held hammer drills of the jackhammer type and heavier plugger machines, drifters of the Leyner type mounted on crossbars or on columns, and drifters (and, now, only occasionally, piston drills) mounted on tripods; more recently, with the more extensive use of bort instead of carbon for diamond drilling, diamond drills are employed occasionally for drilling long blast holes, especially in pillar work.

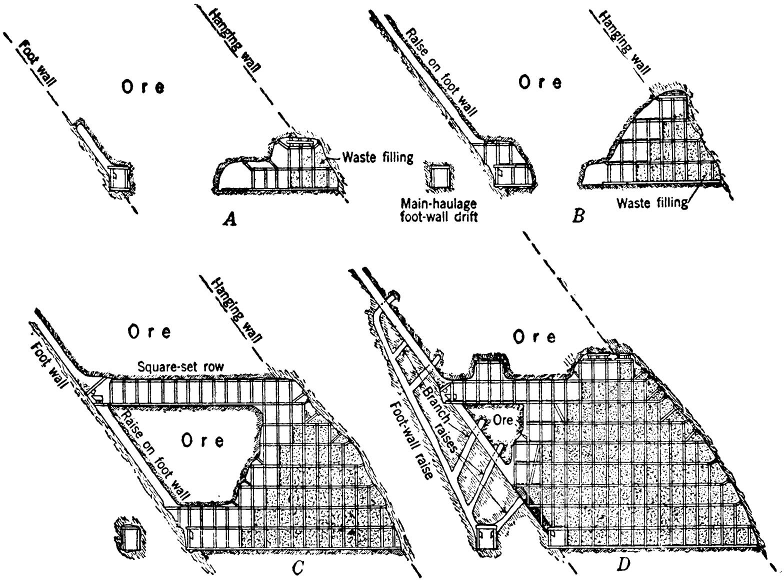
Foot wall
Ore
Hanging wall
Waste filling
A
Main-haulage foot-wall drift
Raise on foot wall
Ore
Hanging wall
Waste filling
B
Ore
Hanging wall
Foot wall
Square-set row
Raise on foot wall
Ore
C
Ore
Ore
Branch raises
Foot-wall raise
D

FIGURE 107.—Square-set system, Bunker Hill and Sullivan mine, Idaho. *A*, Stoping begun at hanging wall and raise started on footwall. *B*, Stoping advancing toward footwall; raise on footwall completed and main haulage advancing in footwall. *C*, Sill floor completely developed and top of stope connected to footwall raise. *D*, First lift of stope nearly completed and footwall and branch raises from haulage drift completed.

For open stoping in flat or slightly inclined deposits not too thick to be mined from floor to back in one slice, light hand-held or mounted drifter machines usually are employed where the ground is easy to drill and heavier mounted drifters are used in harder drilling ground. In thicker deposits a heading-and-bench system of drilling and blasting may be employed, carrying a 6- to 8-foot advance heading or breast under the top of the ore and then benching the bottom ore behind it (fig. 108). Column-mounted drifter machines usually are employed for driving the heading, and bench holes may be drilled with hand-held machines, as at *B* in figure 108. Hand-held machines usually are used for stope holes, as at *A*, and tripod-mounted machines are used for drilling flat "splitter" holes in hard ground, as at *D*. Down holes usually are preferred because hand-held "pluggers" may be employed; it is usually possible to drill a greater footage per machine shift with pluggers if the rock is not too hard, since no time is required for setting up. Under some conditions, however, as in the Tri-State zinc and lead district, where the ground is hard, cherty, seamed, and vuggy, flat holes can be drilled and cleaned out more readily, and fewer holes are lost because of stuck steel. The best system to employ in any particular instance therefore will depend on the nature of the rock. The bench system, using down holes, is also employed in large

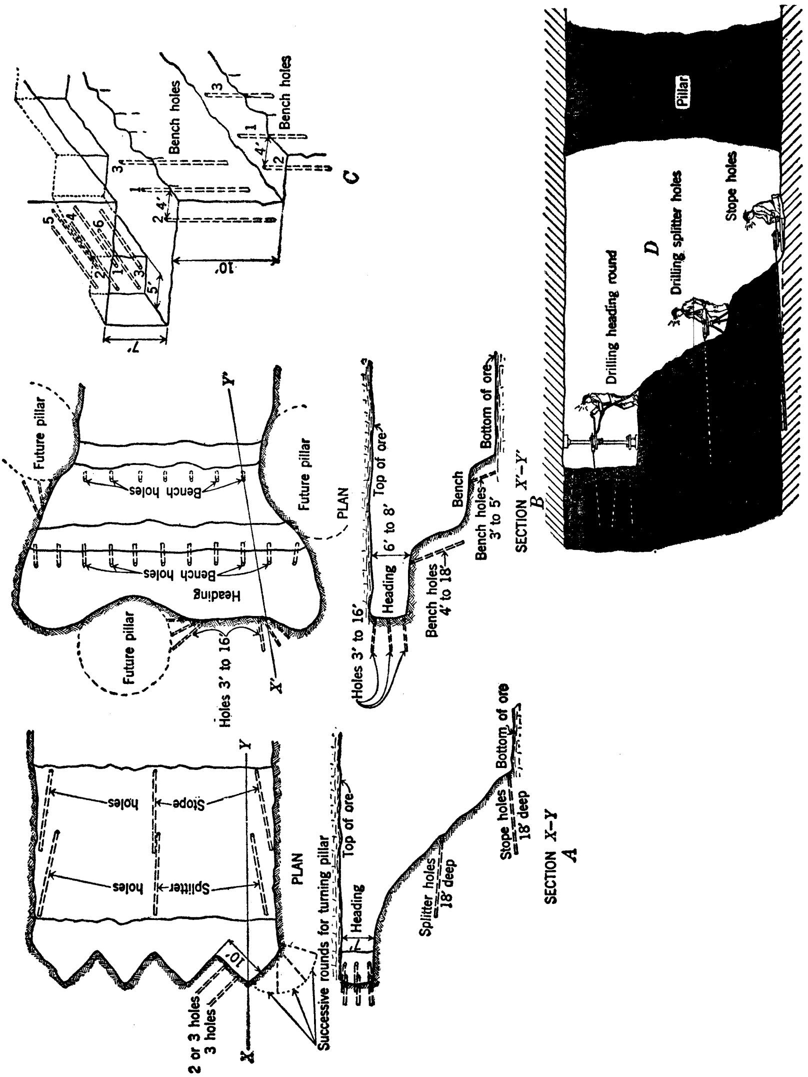
Holes
Stope
Splitter
holes
PLAN
X
Y
2 or 3 holes
3 holes
10'
Successive rounds for turning pillar
Heading
Top of ore
Splitter holes 18' deep
Stope holes 18' deep
Bottom of ore
SECTION X-Y
A
Future pillar
Bench holes
Heading
Holes 3' to 16'
X'
Y'
PLAN
6' to 8'
Bench holes 4' to 18'
Bench
Bench holes 3' to 5'
SECTION X'-Y'
B
Bench holes
1
2
3
4
5
6
4'
2.4'
5'
7'
10'
C
Drilling heading round
Drilling splitter holes
Stope holes
D
Pillar

FIGURE 108.—Details of heading-and-bench stoping. *A*, Stope and splitter holes drilled flat from stope face. *B*, Stope or bench holes drilled nearly vertical from benches. *C*, Bench holes vertical; heading driven in steps. *D*, Drilling heading and stope, Tri-State district.

thick deposits for breaking down ore around a central raise ("milling" or "glory-hole" mining). Figure 108 shows the top of a stope of this type at *C*.

In sublevel stoping the ore is broken partly by slabbing rounds to form the bench, using column-mounted drifters for drilling, and partly by benching (fig. 86), or it may be done by "ring drilling" from and around the sublevel drifts. Bench holes may be drilled upward in the bench above, using stopers or mounted drifters, or pluggers may be employed for drilling downward into the bench below, or holes may be drilled both up and down, as in figure 86. In some mines, only down-hole drilling is done. Where the sublevel interval is large and the benches thick, this may require double or even triple benches, one below the other. By using sectional drill steel, a thick bench may be drilled as a unit. At Flin Flon, Manitoba, bench holes are drilled up to 24 feet in depth.[34] Heavy drifters mounted on a long cross-arm between two 3½-inch columns drill about 40 feet per machine shift in hard, massive sulfide ore. Holes are collared with 3-inch bits and gage reduction is ⅛ inch for each 2 feet of advance.

Drilling practice in shrinkage and cut-and-fill stopes (both overhand systems) is similar. Holes may be drilled vertically or at steep angles upward into the back of the stope with stoper machines, or breasts may be carried in which long, flat holes are drilled with hand-held or mounted hammer drills (fig. 102). The choice between these systems should be governed to a considerable extent by the way in which the ground breaks. This, in turn, may be influenced by a number of factors, prominent among which are the nature and direction of slips, bedding planes, or other planes of weakness in the ore. Another important consideration is the effect that the method of drilling and blasting may have upon the hanging wall and the back of the stope and upon the direction in which the ore is thrown by the blast. Stoper holes are more likely to shatter the back of the stope, and if bottomed close to a wall may shatter and loosen it also. Generally speaking, long, flat breast holes will break more ground per foot of hole and per pound of explosive used than vertical holes, because they have a free underface to break to; also, they will not scatter the broken ore as much as vertical holes. In narrow and moderate-width veins breast holes, if not too long, often can be confined better to the ore, as the trend of the vein can be observed from two sides—the breast and the bottom. There are exceptions to this where sudden changes in the strike of the vein are more pronounced than rolls or changes in dip. In wide stopes the breast-hole system usually is cheaper and often safer because the miner is not standing directly under the ground in which he is drilling. Care must be taken, however, to avoid putting too much burden on the holes, causing the ore to break in large slabs, which will require secondary blasting.

In square-set stoping it usually is desired to break only one set of ground at a time, and care must be taken in pointing and charging the holes to avoid blasting out or breaking the timbers, which are set close to the face. Usually, ground that requires square sets to support the stopes is of such a character that only light blasting is needed, though this is not always the case.

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[34] Marshall, W. J., The Mining Methods of the Flin Flon Mine: Canadian Min. and Met. Bull. 309, January 1938, pp. 31–44.

Either breast stoping with mounted machines or steeply inclined holes drilled with stopers may be employed in overhand square-set stopes. At the Bunker Hill and Sullivan mine stoper drills are preferred (fig. 107, *D*), whereas at the Page mine in the same district breast stoping with light drills is believed to give better control of breaking.

Stopers are employed more widely in square-set stopes than are drifters or mounted jackhammers. As in cut-and-fill and shrinkage stoping, breast holes shatter the back less than stoper holes. When drilling horizontal holes, the miner works from inside a timbered set and is fully protected from falling ground by the timber; whereas, with stopers he is often working directly under the ground in which he

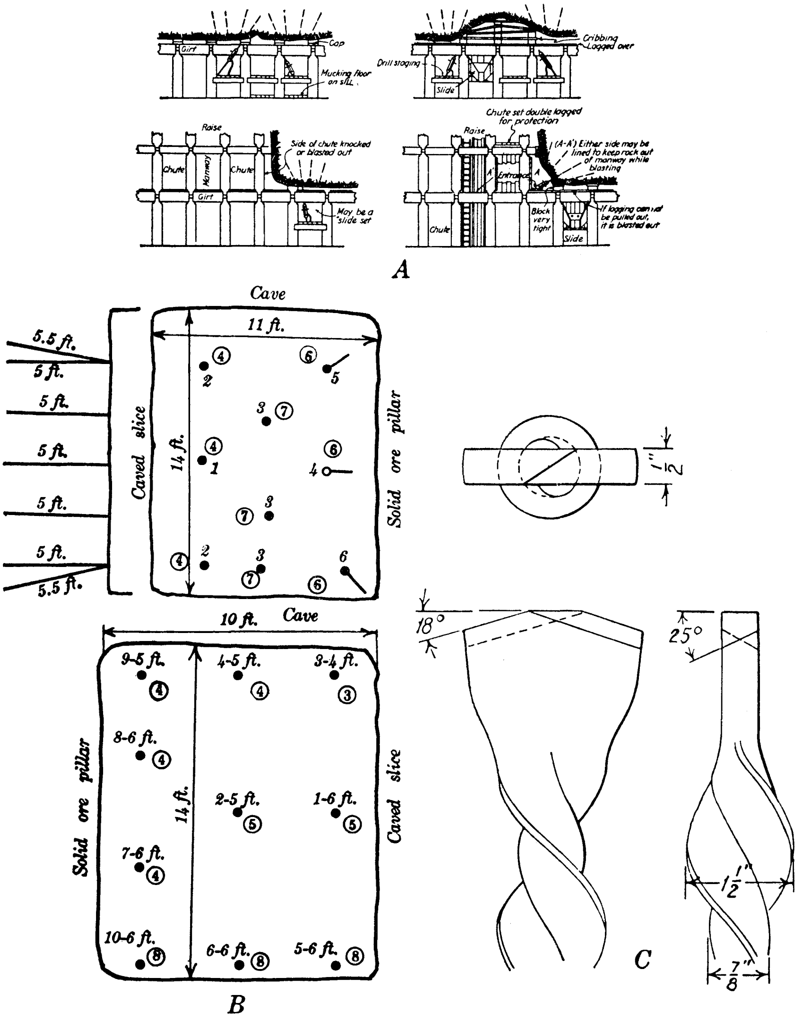
Girt
Cap
Mucking floor on sill
Drill staging
Slide
Cribbing
Logged over
Raise
Chute
Manway
Chute
Girt
Side of chute knocked or blasted out
May be a slide set
Chute set double logged for protection
Raise
(A-A') Either side may be lined to keep rock out of manway while blasting
A'
Entrance
A
Block very tight
If logging cannot be pulled out, it is blasted out
Chute
Slide
A
Cave
11 ft.
14 ft.
5.5 ft.
5 ft.
5 ft.
5 ft.
5 ft.
5 ft.
5.5 ft.
Caved slice
Solid ore pillar
10 ft.
Cave
14 ft.
9-5 ft.
4-5 ft.
3-4 ft.
8-6 ft.
2-5 ft.
1-6 ft.
7-6 ft.
10-6 ft.
6-6 ft.
5-6 ft.
Solid ore pillar
Caved slice
B
1/2"
18°
25°
1 1/2"
7/8"
C

FIGURE 109.—*A*, Drilling with stopers in square-set stopes. *B*, Top-slice rounds in soft ore. *C*, Auger bit on twisted steel.

is drilling vertical holes, or only has partial protection when drilling inclined holes (fig. 109, *A*).

When stopers are used, no time is lost rigging up on mountings, but in some cases this may be more than offset by the necessity of rigging staging. Several methods of drilling in square-set stopes with stoper machines are shown in figure 109, *A*.

Steel breakage is generally greater with stopers than with mounted machines, and in hard ground automatically rotated stopers entail greater maintenance and repair expense.

In block-caving, most of the ore is broken by natural forces and drilling is confined to development work (driving of boundary or cut-off drifts, raises, and shrinks), to undercutting operations, and to bulldozing boulders on the grizzly levels. Mounted machines or hand-held jackhammers are used in boundary drifts and stopers in boundary raises and shrinkage stopes. Pluggers are employed for funneling raises on the undercutting level; where undercutting is done by checkerboarding with a series of drifts and blasting out the pillars, hand-held jackhammers are used for drilling the pillars and stopers for drilling the backs of the drifts. Ground suitable for block-caving is usually easily drilled, and mounted machines seldom are required in undercutting operations.

Top-slicing in soft ore requires little drilling, and only a few holes are required to break a 5- or 6-foot round in a large slice (fig. 109, *B*). Soft ores often can be drilled by hand with auger steel fashioned with a fishtail bit. In recent years light, hand-held machines and auger bits (fig. 109, *C*) often are employed. In harder ground mounted or unmounted machines and standard steel and bits are employed to drill regular rounds similar to drift and crosscut rounds previously discussed under Development. Rounds in sublevel caving slices are similar to those for top-slicing under comparable ground conditions.

Stope drilling often is not as efficient as it might be; actual drilling time commonly occupies but 2½ to 3½ hours during an 8-hour shift, even when much more drilling could be done if there were time for it. This is due partly to the necessity of barring down the back and making it safe before drilling, an operation that may require 1 to 2 hours. In some stopes, drills, mountings, and hose have to be dragged in over a rough, uneven muck pile and connected and machines rigged up for drilling (when mounted machines are employed) or staging installed. In timbered stopes much of the shift may be used for timbering. Upon completion of the round, drills, steel, and gear have to be removed and the holes charged and blasted. All these operations are essential, and the resultant loss of drilling time is inherent in some stoping methods. With other methods, such as sublevel stoping and underhand bench or "glory-hole" stoping or in drilling pillars, it is often possible to drill for several shifts before blasting, thus increasing the ratio of actual drilling time to total time. Even with shrinkage, cut-and-fill, or square-set stoping, by making changes in practice to permit drilling over a considerable area before blasting, the actual drilling time per shift often can be increased.

Delays occasioned by shortage of drill steel, taking steel into the stopes, going out during the shift for hose or pipe couplings or other small items of equipment or for tools are common in many mines. Such delays often can be reduced by more careful attention on the part of the foremen and stope bosses to distribution of materials and supplies or by providing better facilities for taking timber and supplies into the stopes.

Where detachable drill bits have been found suitable for drilling, their use may save much time in handling drill steel. One company that uses about 6,500 steels per day expects to save $50,000 per year in "nipping" charges alone by susbtituting detachable bits for conventional forged-steel bits.

Published data correlating speed of drilling or rate of penetration, actual drilling time, and tons of ore broken per machine shift in stopes are meager. However, a few typical data on feet drilled per shift and tons of ore broken per machine shift are given in table 33.

Considerable dry drilling is still done in some mines, but this practice is dangerous to health and should be discouraged.

TABLE 33.—*Data on drilling and blasting in stopes*

Mine	Stoping method	Average stope width, feet	Drilling		Tons ore broken per machine shift
			Machines and mountings	Drilling per machine shift, feet	
Mineville, N. Y.[1]	Open stopes, casual pillars.	30±	Light, hand-held		41.06.[2]
Mine No. 1, Tri-State, district.[3]	Open stopes, casual pillars; heading and bench.	50±	Heavy drifters on columns or tripods.		40–60.
Marquette range, Mich.[4]	Open stopes, regular pillars, breast stoping.	25	3½-inch drifters on tripods.	25–30	25–30.[2]
Burra-Burra, Tenn.[5]	Sublevel stoping	40	Mounted drifters for slabbing, jackhammers in bench.	90 (slabbing)	48.
Waco, Mo.[6]	Open stopes, casual pillars, heading and bench.	40–60	Heavy drifters on columns or tripods.		30–40 in headings, 80–100 in benches, av. 62.
Barr, Kans.[7]	do	30	do		64.2.
Hartley, Tri-State district.[8]	do	40	do	60	62.3.
Southeast Missouri.[9]	do	30	Jackhammers on pneumatic leg.	80–100	40.
Fresnillo, Mexico, hilltop ore body.[10]	Surface glory holes, benching around raises.	10 to wide	55-lb. jackammers, hand-held.	120	130+.
Mt. Isa, Queensland.[11]	Sublevel stoping by "ring drilling."	30	4-inch mounted drifters and 3-inch stopers.	60–90	17.5.
Do.[11]	Sublevel stoping by benching.	6–14	3½-inch mounted drifters and "pluggers."		

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[1] Cummings, A. M., Method and Cost of Mining Magnetite in the Mineville District, N. Y.: Bureau of Mines Inf. Circ. 6092, 1928, 12 pp.

[2] Long tons.

[3] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.: Bureau of Mines Inf. Circ. 6113, 1929, 11 pp.

[4] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[5] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[6] Banks, L. M., Mining Methods and Costs in the Waco District: Bureau of Mines Inf. Circ. 6150, 1929, 10 pp.

[7] Keener, O. W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

[8] Anderson, C. N., Mining Methods and Costs at the Interstate Zinc & Lead Co.'s Hartley Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6656, 1932, 16 pp.

[9] Jackson, Chas. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Bureau of Mines Inf. Circ. 6170, 1929, 21 pp.

[10] Livingston, A., Mining Methods and Costs at Fresnillo, Zacatecas, Mexico: Bureau of Mines Inf. Circ. 6661, 1932, 31 pp.

[11] Kruttschnitt, J., and Mann, V. I., Mining Methods and Costs at Mount Isa Mines, Ltd., Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

TABLE 33.—*Data on drilling and blasting in stopes*—Continued

Mine	Stoping method	Average stope width, feet	Drilling		Tons ore broken per machine shift
			Machines and mountings	Drilling per machine shift, feet	
Engels, Calif.[12]	Shrinkage	12–50	Mounted drifters	56.2	27.0.
Do.[12]	Sheet pillars	12–50	do	56.2	75.0.
Do.[12]	Vertical pillars	12–50	Diamond drill		
Nevada-Massachusetts, Nevada.[13]	Shrinkage	4.5	Stopers in soft ground; drifters in hard ground.	70 60	11.4.
Mt. Hope, N. J.[14]	do	19	Jackhammer fitted to stoper leg, mounted on wooden plank.	32	82.53 (1930).
Block P, Mont.[15]	Horizontal cut-and-fill.	3–5	Hand rotated stopers.	75	4.88.[16]
Braden, Chile[17]	Shrinkage stopes with pillar caving.	16.4	Stopers	90–120	70.
Bessemer, Ala.[18]	Open stopes	30	Drifters on tripods	125–165	100–125.[2]
Alan Wood, N. J.[19]	Shrinkage	30	Stopers		69 (1931).[2]
Do	Sublevel stoping		Mounted drifters; jackhammers for vertical holes.		92 (1931).[2]

[2] Long tons.

[12] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[13] Heizer, O. F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Mines at Mill City, Nev.: Bureau of Mines Inf. Circ. 6284, 1930, 13 pp.

[14] Sweet, J. R., Mining Methods and Costs at the Mount Hope Mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp.

[15] Vanderburg, W. O., Mining Methods at the Block P Mine of the St. Joseph Lead Co., Hughesville, Mont.: Bureau of Mines Inf. Circ. 6416, 1931, 14 pp.

[16] Includes sorting.

[17] Webb, J. S., and Skinner, T. W., Mining Methods and Costs at the Braden Copper Co.'s Mines, Sewell, Chile: Bureau of Mines Inf. Circ. 6565, 1932, 13 pp.

[18] De Sollar, T. C., The Red Ore Mines of the Woodward Iron Company at Bessemer, Ala.: Trans. Am. Inst. Min. and Met. Eng., vol. 109, 1934, pp. 69-77.

[19] Loux, C. H., Mining Methods and Costs at the Washington Mine of the Alan Wood Mining Company: Trans., Am. Inst. Min. and Met. Eng., vol. 109, 1934, pp. 51-60.

Blasting in stopes is designed not only to break the ore from the solid but to obtain its proper fragmentation, and the explosive should be selected and the holes loaded with this in mind. Methods of pointing and loading the holes that result in throwing down large slabs or blocks of ore without proper fragmentation are not efficient. Unless the ore is broken small enough by the blast to pass the grizzlies and ore chutes, secondary breaking by plugging and reblasting boulders or by sledging must be resorted to. "Mud-capping" of boulders is sometimes employed but is not considered good practice. Blasts that break the ore into large slabs are also more likely to loosen other slabs and so produce a dangerous condition.

As a rule, blasting in stopes is done at the end of the shift. Where the second shift follows the first immediately, blasting is done only at the end of the second shift; there is then a period of several hours for smoke to clear and dust to settle before men reenter the stope. Regulations at some mines provide that blasting shall be done only at the end of the shift. At others, great loss of efficiency would result from such a rule. In any event, where blasting must be done during the shift, adequate ventilation for quick removal of smoke and dust should be provided. Secondary blasting on grizzlies usually requires that blasting be done during the shift, whereas bulldozing of blocks in the stopes usually can be done at the end of the shift, although occasionally it may be done at lunchtime.

Gelatin dynamites usually are employed for stope blasting because of their low production of noxious gases, and ammonia dynamites frequently are employed for dry work. Straight dynamites should never be employed in confined stopes, and it is best not to use them underground even under most favorable conditions of ventilation on account of the fume produced.

Explosives of 25- to 60-percent equivalent (based on straight dynamite) strength are employed, 40-percent strength being by far the most common. Detonation is by fuse and cap or electric detonators. Most mines cling to the fuse and cap except for large blasts—that is, blasts in which a large number of holes are shot simultaneously or in rapid sequence.

The sequence of firing many shots in a single blast can be controlled more certainly with electric delay denotators than with fuse and cap, and no fume is produced by the firing medium, as there is with fuse. For firing a large number of shots simultaneously, as in blasting out posts in a top slice, or firing a single row of holes around a bench, some favor the use of Cordeau.

Under some conditions space blasting is used to obtain better fragmentation, as in cut-and-fill stopes in massive sulfides at Cananea.[35] The use of 40-percent explosive gave a heaving action that threw down many huge boulders, and substitution of 60-percent explosive helped this somewhat. Results of space blasting with 60-percent explosive were still better. In loading the holes, individual sticks of explosive were separated by 1-inch round wood sticks 10 inches in length. Thus, a stick of explosive was followed by a wood stick, on top of which the next stick of explosive was tamped, and so on. The hole was then sealed with two or three sticks of stemming. Since the wood sticks were only 1 inch in diameter and the hole diameter was 1⅝ to 2 inches, detonation was complete, yet the force of the explosion was distributed along the hole instead of being concentrated in the bottom.

A similar result may be achieved by employing an explosive of low bulk strength, using more cartridges distributed along the length of the hole rather than fewer sticks of high bulk strength concentrated in the lower part of the hole.

Fragmentation, of course, can be improved in many instances by spacing the holes more closely, drilling more holes, and loading lighter, but the additional cost of drilling must be balanced against the additional cost of secondary breaking with fewer holes.

In cut-and-fill stoping it usually is necessary to lay a tight flooring on top of the fill before blasting to keep the ore from mixing with the filling material. This becomes especially important if the ore is high-grade or if the mineral is concentrated in the fines.

In deposits where square-set stoping must be employed, the ground usually is of such nature that comparatively light blasting is required for fragmentation, and the same is true of soft-ore mines employing top-slicing and sublevel caving. Care must be taken, however, to avoid blasting out or breaking the timber, as previously pointed out. Often it is necessary to brace the timbers to prevent blasting them out and to cover the floor (in square-set stopes) with loose ore or lagging to prevent breakage by chunks of ore. In square-set stoping, holes usually are placed and loaded to take advantage of seams and slips in the ore rather than according to a standard system.

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[35] Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Bureau of Mines Inf. Circ. 6247, 1930, pp. 23–24.

The use of stemming in all holes is advocated by explosives manufacturers to achieve better detonation and greater efficiency, and to abate fume production. Tests made some years ago by the Bureau of Mines indicate that "The use of stemming helps decidedly to prevent formation of boulders, does not increase the cost of blasting perceptibly * * *. The tests show that one stick of stemming to a hole is not enough and that at least two should be used."[36] Further tests showed that stemming will almost always result in some saving of powder, and less poisonous gas is produced when it is used.[37] At present (December 1938) exhaustive tests are being made at the Mount Weather testing adit of the Bureau of Mines to check the earlier work. The extra danger and difficulty inherent in firing missed holes when employing stemming partly offset the advantages claimed for is use. Whatever the virtues of stemming may be, the fact remains that most stope holes are loaded without stemming and, in the judgment of many operators, the effect of stemming on both blasting efficiency and gas reduction is small.

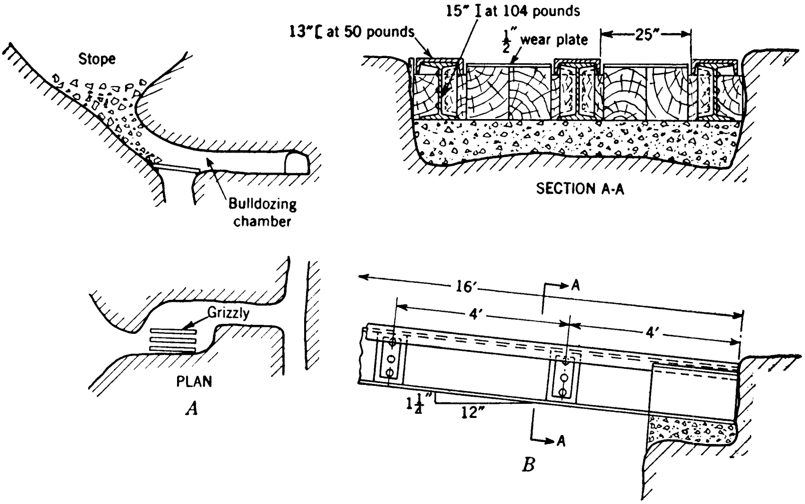

FIGURE 110.—*A*, Bulldozing chamber. *B*, Grizzly details, Alaska-Juneau mine.

Considerable secondary blasting is required in connection with block caving, forced caving, and sublevel stoping in hard ore. It is usually done on grizzlies or in bulldozing chambers situated between the undercutting level and haulage level (see figs. 90 and 97). Blasting directly on the grizzlies is destructive to the grizzly bars and settings, and the recent trend is toward blasting in chambers on one side of the grizzly and transferring the reblasted ore by scrapers to the grizzlies.

Figure 110 shows a bulldozing chamber and details of grizzly construction at the Alaska-Juneau mine in 1928.[38] In that year, 71 percent of the explosive consumed was used in secondary breaking, 13 percent in development, and 16 percent in stope blasting (in powder drifts, see fig. 89).

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[36] Gardner, E. D., and Howell, S. P., Blasting to Lessen Bowlders in Hard-Ore Stopes: Bureau of Mines Tech. Paper 383, 1926, pp. 22–23.

[37] Gardner, E. D., Howell, S. P., and Jones, G. W., Gases from Blasting in Tunnels and Metal-Mine Drifts: Bureau of Mines Bull. 287, 1927, p. 93.

[38] Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Bureau of Mines Inf. Circ. 6186, 1929, 18 pp.

In shrinkage and cut-and-fill stoping, large boulders usually are reblasted in the stope. If they become covered with finer muck in shrinkage stopes they may be missed, however, and finally show up at the chutes, where blasting may result in injury to the chute gates and timbers.

Boulders may be either block-holed by drilling with hand-held machines (or stopers, if boulders are wedged behind the brow of a draw raise) and loaded with a light charge and blasted; or they may be broken by "mud-capping," which consists in placing a charge of explosive on top of the rock to be blasted and plastering it down with clay or wet fine material.

Block-holing is the approved method, from a safety standpoint, and powder consumption usually is appreciably less than when breaking is done by mud-capping. Furthermore, if several mud-capped boulders are blasted together the concussion from the first one to go may throw the charge off one or more of the others.

Data on explosives consumption in stoping are presented in table 34.

HANDLING ORE IN STOPES

With some methods of stoping, the broken ore moves by gravity directly to the loading chutes on the haulage level without any handling from the time it is blasted until it is drawn from the chutes.

Thus, in shrinkage stoping the ore is drawn from the bottom of the column of broken ore that fills the stope, through closley spaced chutes on the haulage level. In sublevel stoping (fig. 86) the ore is blasted out into an open stope and falls into funneled raises or mill holes, and no handling is needed except where benching down is done in two or more steps, when some shoveling may be necessary to clean off the lower benches. With block-caving, most of the ore broken in undercutting falls by gravity into raises previously driven at close intervals, and very little shoveling is required. After caving starts, the stopes are not entered, of course, and the caved ore is drawn by gravity through draw sets or finger raises and is passed through grizzlies to the loading chutes. Thus, there is no handling of ore in the stope proper. In underhand benching to mill holes considerable hand-shoveling may be required to clean off the lower benches after blasting.

Cut-and-fill and square-set stopes often are carried on an angle slightly steeper than the angle of repose of the broken ore (rill stoping) so that it will run by gravity to the chutes and thus a minimum of shoveling be required.

In flat-back (horizontal) cut-and-fill stopes, square-set stoping with horizontal floors, top-slicing, and sublevel caving some ore handling must be done in the stopes, although the amount may be reduced by providing closely spaced ore passes. The cost of building and maintaining a large number of ore passes may be greater, however, than that of wheeling, tramming, or scraping the ore in the stopes, especially in high stopes in heavy ground.

In mining tabular deposits dipping at angles flatter than the angle of repose of the broken ore, it becomes necessary either to drive drifts in the footwall and put up rock raises to the stopes for gravity handling or to adopt some means for moving the broken ore down the dip. The former entails high preparatory expense for deadwork, and since the development of relatively inexpensive and very efficient

TABLE 34.—*Explosive consumption in stoping*

I. OPEN STOPING

Mine	Variation of stoping method	Kind and grade of explosive	How fired [1]	Stemming	Explosive per ton ore broken, pounds	Remarks
Tri-State No. 1 [2]	Casual pillars, heading, and bench.	Ammonia, 33 percent		Clay cartridge	0.750	Bench holes sprung twice.
Tri-State No. 2 [3]	do	Ammonia, 33 percent; primers, 30 percent gelatin.	Fuse and No. 6 caps	do	1.265	Do.
Waco, Mo. [4]	do	Bulk, 15 and 25 percent	do	Tailings	.594	
Barr, Kans.[5]	do	Ammonia, 40 percent	Fuse and No. 8 caps	None	.805	
Tri-State No. 3 [6]	Casual pillars, breast stoping	Gelatin dynamite, 20 percent.	Fuse and No. 6 caps	?	.875	Holes sprung twice.
Hartley-Grantham, Tri-State district.[7]	Casual pillars, heading, and bench.	Gelatin dynamite, 30 percent.	Fuse and No. 8 caps	None	.741	Maximum depth of holes 18 feet.
Hartley, Tri-State district [8]	do	Ammonia, 40 percent; primers, 35 percent; gelatin dynamite.	Fuse and No. 6 caps	Tailing sludge	1.120	Primers inserted in cardboard tubes.
No. 8, Southeast Missouri [9]	do	Bulk gelatin, 35 percent; ammonia gelatin, 35 percent.	do		.565	75 percent bulk and 25 percent ammonia gelatin.
Lead mine, Southeast Missouri.[1]	do	Ammonia gelatin, 40 percent.	do	Drill cuttings in cartridges.	.459	
Mascot, Tenn.[11]	Casual pillars, mill-hole benching system.	Gelatin dynamite, 30 percent.			.502	
Marquette No. 1, Mich.[12]	Regular stope and pillar; breast stoping.	L. F. ammonia, 50 percent.			[13] .705	
Menominee No. 1, Mich.[14]	Sublevel stoping	Ammonia, 40 percent			[13] .619	
Iron mine A	do	Gelatin dynamite, 40 percent.			[13] .610	
Iron mine B	do	do			[13] .674	
Iron mine D	do	do			[13] .500	
Burra-Burra, Tenn.[15]	Sublevel stoping; all down holes.	do	Electric		[16] .755	Bulldozing, 0.167 pound per ton included.
Montreal, Wis.[17]	Sublevel stoping; all up holes.	Ammonia, 35 and 60 percent.			[13] .490	Overhand holes chambered.
Hanover-Bessemer, N. Mex.[18]	Underhand benching to inclined raises.				[13] .451	
Mineville, N. Y.[19]	Casual pillars, breast, and heading and bench.	Gelatin dynamite, 25 and 40 percent.			[13] .799	
Mary, Tenn.[20]	Regular pillars, underhand benching.	Gelatin dynamite, 40 percent.			.420	
Vanadium, Colo.[21]	Regular pillars, breast stoping.	Special bulk dynamite			1.050	Thin ore body.

El Potosi, Mexico [22]	Underhand glory hole	Gelatin dynamite, 35 percent.	No. 6 caps and fuse or electric.		.384	
Mount Isa, Queensland [23]	Sublevel stoping, "ring drilling."	Gelatine dynamite, 40 and 60 percent.	Electric		.427	Space blasting with 1¼- by 8-inch wooden spacers.

II. SHRINKAGE STOPING

Harmony, Idaho [24]	On drift pillars; stoper holes	Gelatin dynamite 35 percent.	?	?	2.05	Considerable secondary breaking.
Engels, Calif.[25]	On drift pillars; breast holes	Ammonia dynamite 40 percent; some 25, 60, and 90 percent.	Electric 220-volt circuit.	Sawdust and clay cartridges.	.813	Includes some pillar blasting with long holes.
Black Butte, Oreg.[26]	do	Gelatin dynamite 35 percent.	Fuse and No. 6 caps		1.16	Ore 20 feet wide.
Nevada-Massachusetts, Nev.[27]	On pillars; both stoper and breast holes.	do	do	None	1.650	Ore averages 4½ feet thick.
Cortez, Nev.[28]	On pillars; breast holes	Gelatin dynamite 40 percent.			2.030	Average ore width 7 feet.
Marquette No. 4, Mich.[29]	On pillars; breast holes; semishrinkage between pillars.	Gelatin dynamite 50 percent.			[13].670	Stopes 20 to 25 feet wide.
Verde Central, Ariz.[30]	On stulls; both stoper and breast holes.	Gelatin dynamite 40 percent; some 60 percent in hard ore.	Fuse and No. 8 caps		1.160	

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[1] Unless otherwise indicated, blasting probably with fuse and caps.

[2] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.: Bureau of Mines Inf. Circ. 6113, 1929, 11 pp.

[3] Netzeband, W. F., Method and Cost of Mining Zinc and Lead Ore at Mine No. 2, Tri-State District, Picher, Okla.: Bureau of Mines Inf. Circ. 6121, 1929, 11 pp.

[4] Banks, L. M., Mining Methods and Costs in the Waco District: Bureau of Mines Inf. Circ. 6150, 1929, 10 pp.

[5] Keener, O. W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

[6] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 3 Mine, Tri-State District, Crestline, Kans.: Bureau of Mines Inf. Circ. 6174, 1929, 10 pp.

[7] Keener, O. W., Methods and Costs of Mining at the Hartley-Grantham Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6286, 1930, 8 pp.

[8] Anderson, C. N., Mining Methods and Costs at the Interstate Zinc & Lead Co.'s Hartley Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6656, 1932, 16 pp.

[9] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

[10] Jackson, Chas. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Bureau of Mines Inf. Circ. 6170, 1929, 21 pp.

[11] Coy, H. A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Bureau of Mines Inf. Circ. 6239, 1930, 11 pp.

[12] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[13] Per long ton (2,240 pounds).

[14] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.: Bureau of Mines Inf. Circ. 6180, 1929, 10 pp.

[15] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[16] Includes some development, shrinkage stoping, and pillar mining.

[17] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

[18] Kniffin, L. M., Mining and Engineering Methods and Costs of the Hanover-Bessemer Iron & Copper Co., Fierro, N. Mex.: Bureau of Mines Inf. Circ. 6361, 1930, 20 pp.

[19] Cummings, A. M., Method and Cost of Mining Magnetite in the Mineville District, New York: Bureau of Mines Inf. Circ. 6092, 1929, 12 pp.

[20] Kegler, V. L., Mining Methods of the Ducktown Chemical & Iron Co., Mary Mine, Isabella, Tenn.: Bureau of Mines Inf. Circ. 6397, 1931, 9 pp.

[21] Burwell, Blair, Mining Methods and Costs at the Vanadium Mine of the United States Vanadium Corporation, Rifle, Colo.: Bureau of Mines Inf. Circ. 6662, 1932, 9 pp.

[22] Walker, Harlan A., Mining Methods and Costs at El Potosi Mine, El Potosi Mining Co., Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6804, 1934, 38 pp.

[23] Kruttschnitt, J., and Mann., V. I., Mining Methods and Costs at the Mount Isa Mines, Ltd., Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

[24] Gardner, E. D., Mining Practice at Harmony Mines Co., Baker, Idaho; Bureau of Mines Inf. Circ. 6240, 1930, 8 pp.

[25] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

[26] Elmer, W. W., Mining Method and Costs at the Black Butte Quicksilver Mine, Lane County, Oreg.: Bureau of Mines Inf. Circ. 6276, 1930, 8 pp.

[27] Heizer, O. F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Mines at Mill City, Nev.: Bureau of Mines Inf. Circ. 6284, 1930, 13 pp.

[28] Hezzelwood, G. W., Mining Methods and Costs at the Consolidated Cortez Silver Mines, Cortez, Nev.: Bureau of Mines Inf. Circ. 6327, 1930, 15 pp.

[29] Graff, W. W., Mining Practices, Methods, and Costs, Mine No. 4 of the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6390, 1930, 8 pp.

[30] Dickson, R. H., Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.: Bureau of Mines Inf. Circ. 6464, 1931, 13 pp.

TABLE 34.—*Explosive consumption in stoping*—Continued

II. SHRINKAGE STOPING—continued

Mine	Variation of stoping method	Kind and grade of explosive	How fired	Stemming	Explosive per ton ore broken, pounds	Remarks
Kirkland Lake, Ontario [31]	On stulls; breast holes	Gelatin dynamite 40 percent.	do		2.540	
Vipond, Ontario [32]	On timbers and on pillars; breast holes.	do			1.050	Pillars in wide or lean ore. On timbers in narrow good ore.
Elkoro, Nev.[33]	On stull timbers; stoper holes	do			1.239	Average width vein, 6 feet.
Mount Hope, N. J.[34]	On pillars; breast holes	Gelatin dynamite 40 and 50 percent.	Fuse and No. 8 caps	Yes	.314	14-foot holes with wood spacers in top half.
Cresson, Colo.[35]	On pillars; stoper holes	Gelex No. 2	Fuse and No 6 caps		2.300	Mostly narrow viens.
Hog Mountain, Ala.[36]	Usually on pillars; stoper holes.	Gelatin dynamite 60 percent.	Electric		1.500	
Eustis, Quebec [37]	On pillars; breast holes	Gelatin dynamite 40 percent.	Fuse and No. 8 caps		.763	Holes up to 12 to 15 feet long.
United Verde, Ariz.[38]	Horizontal on pillars	Gelatin dynamite 50 percent.	do		.710	

III. CUT-AND-FILL STOPING

Mine	Variation of cut-and-fill stoping	Kind and grade of explosive	How fired	Explosives, pounds per ton broken	Remarks
Matahambre, Cuba [39]	Flat back on drift pillars	Gelatin dynamite 30 percent	Fuse and No. 8 caps	0.44	
Hecla, Idaho [40]	Stringer set and fill	do		.533	
Morning, Idaho [41]	do	Gelatin dynamite 25, 35, 40 percent.		.693	
Pilares, Mexico [42]	Horizontal			.415	On level or on drift pillars.
	Inclined			.345	On drift pillars.
Pecos, N. Mex.[43]	Horizontal on drift pillars	Gelatin dynamite 25 and 40 percent.		.85	(1928.)
				.73	(1929.)
Block P, Mont.[44]	Horizontal resuing in narrowest ore	Gelatin dynamite 30 percent	Fuse and No. 6 caps	2.568	Vein narrow.
United Verde, Ariz.[45]	Horizontal on pillars	Gelatin dynamite 50 percent	Fuse and No. 8 caps	.55	30 percent explosive for bulldozing consumes 20 percent of total.
Questa, N. Mex.[46]	Horizontal on timbers	Gelatin dynamite 40 percent	Fuse and No. 6 caps	2.98	Includes some explosive used in development.
Champion, Mich.[47]	Sublevel inclined on timbers		Fuse and cap	.80	
Cold Springs, Colo.[48]	Flat back on stulls. Resuing in narrow ore.	Gelatin dynamite 40 percent	do	[49] 4.48	

Teziutlan, Mexico [50]	Horizontal on drift pillars	do	do	.947	
McIntyre, Ont.[51]	Horizontal on timbers	do	do	.870	Some shrinkage and a little square-set stoping.
Atlanta, Idaho [52]	do	Gelatin dynamite 30 percent	do	.148	Some square-setting in widest ore.
Park City Consolidated, Utah [53]	Inclined on timbers	do	do	.552	

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[31] Dumbrille, J. C., Mining Methods of Kirkland Lake Gold-Mining Co. (Ltd.), at Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6490, 1931, 12 pp.

[32] Dye, R. E., Mining Practices and Costs of the Vipond Mine Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, 11 pp.

[33] Park, J. F., Mining Practices, Methods, and Costs at Elkoro Mines, Jarbidge, Nev.: Bureau of Mines Inf. Circ. 6543, 1931, 12 pp.

[34] Sweet, J. R., Mining Methods and Costs at the Mount Hope Mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp.

[35] Beebe, A. H., and Johnson, C. H., Mining Methods and Costs at the Cresson Mine, Cripple Creek, Colo: Bureau of Mines Inf. Circ. 6805, 1934, 14 pp.

[36] Johnson, N. O., Mining and Milling Methods and Costs, Hog Mountain Gold Mining & Milling Co., Alexander City, Ala.: Bureau of Mines Inf. Circ. 6914, 1936, 23 pp.

[37] Snow, Fred W., and Brownbill, H. F., Mining Methods and Costs at the Eustis Mine: Canadian Min. and Met. Bull. 286, February 1936, Transactions Sec., pp. 70–85.

[38] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[39] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

[40] Foreman, C. H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Bureau of Mines Inf. Circ. 6232, 1930, 21 pp.

[41] Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6238, 1930, 13 pp.

[42] Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares, Sonora, Mexico: Bureau of Mines Inf. Circ. 6307, 1930, 34 pp.

[43] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 25 pp.

[44] Vanderburg, W. O., Mining Methods at the Block P Mine of the St. Joseph Lead Co., Hughesville, Mont.: Bureau of Mines Inf. Circ. 6416, 1931, 14 pp.

[45] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[46] Garman, J. B., Mining Methods of the Molybdenum Corporation of America, at Questa, N. Mex.: Bureau of Mines Inf. Circ. 6514, 1931, 15 pp.

[47] Mendelsohn, Albert, Mining Methods and Costs at the Champion Mine, Painesdale, Mich.: Bureau of Mines Inf. Circ. 6515, 1931, 16 pp.

[48] Vanderburg, W. O., Methods and Costs of Mining Ferberite Ore at Cold Springs Mine, Nederland, Boulder County, Colo.: Bureau of Mines Inf. Circ. 6673, 1932, 15 pp.

[49] Based on sorted ore. Ore broken was 6.51 times the tonnage of sorted ore; thus the explosive consumption per ton broken was 0.69 pound.

[50] Herivel, E. Ph., Mining Methods and Costs at the Teziutlan Copper Mine of the Mexican Corporation, S. A., Teziutlan, Puebla, Mexico: Bureau of Mines Inf. Circ. 6736, 1933, 16 pp.

[51] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, 19 pp.

[52] Gardner, E. D., Mining Methods and Costs at the Mine of the St. Joseph Lead Co., Atlanta, Idaho: Bureau of Mines Inf. Circ. 6823, 1935, 7 pp.

[53] Wiles, Gloyd M., Mining Methods and Costs at the Park City Consolidated Mines Co., Park City, Utah: Bureau of Mines Inf. Circ. 6880, 1936, 13 pp.

TABLE 34.—*Explosive consumption in stoping*—Continued

IV. SQUARE-SET STOPING

Mine	Variation of square-set stoping	Kind and grade of explosive	How fired	Stemming	Explosives, pounds per ton broken	Remarks
United Verde Extension, Ariz.[54]	Vertical sections		Fuse and caps		0.39	
Argonaut, Calif.[55]	Horizontal floors	Gelatin dynamite 30 percent	Fuse and No. 6 caps		1.01	Includes some powder used in development.
Tintic Standard, Utah [56]	Vertical sections	Gelatin dynamite, half 25 percent, half 40 percent.			.418	
Ground Hog, N. Mex.[57]	Rill				.63	
Central Eureka, Calif.[58]	Horizontal floors	Gelatin dynamite, 40 percent	Fuse and No. 6 caps		1.01	
Talache, Idaho [59]	do				.77	
Bawdwin, Burma [60]	Rill	Gelignite 50 percent	Fuse and No. 7 caps		.36	

V. BLOCK-CAVING

Mine	Kind and grade of explosive	How fired	Stemming	Explosive, pounds per ton broken	Remarks
Morenci, Ariz.[61]	Ammonia, 30 percent for block-holing. Gelatin dynamite, 40 percent in undercutting.	Mostly electric	Sand-filled cartridges	0.24 (1927) .19 (1928)	About 63 percent of total for block-holing.
Ray, Ariz.[63]	Gelatin dynamite, 30 percent	Mostly fuse and cap		.252 (1927) .319 (1928)	
Braden, Chile [63]	Ammonia gelatin, 25 percent			.035 .038	Undercutting. Chute and ore-pass blasting.
Miami, Ariz.[64]	Gelatin dynamite, 40 percent	Electric for chute blasting		.1725 .0500	Mining. Chute-blasting.

VI. SUBLEVEL CAVING AND TOP-SLICING

Mine and reference	Variation of stoping method	Kind and grade of explosive	How fired	Explosive, pounds per ton broken	Remarks
Eureka-Asteroid, Mich.[65]	Sublevel caving	Gelatin dynamite, 40 percent	Fuse and caps	0.382	
Montreal, Wis.[66]	do	Ammonia gelatin, 35 percent	do	.394	1928.
Do.[67]	Sublevel caving, large-pillar method	Gelatin dynamite, 40 percent	Fuse and No. 6 caps	.625	1937.
Mine 16 [67]	Sublevel caving and top-slicing	Gelatin dynamite, 50 percent	Fuse and caps	[69].421	

Mine 20 [68]	do	Ammonia gelatin, 40 percent	do	[70] .553
Mine 1, Minn.[71]	Parallel top-slicing	Ammonia gelatin, 35 percent, and special gelatin, 40 percent.	do	[69] .395
Mine 2 [68]	do	Special gelatin, 35, 40, and 60 percent	do	[70] .601
Mine 3 [68]	do	Special gelatin, 40 percent	do	[70] .694
Mine 4 [68]	do	Gelatin dynamite, 35 percent	do	[70] .463
Mine 5 [68]	do	Special gelatin, 30 percent	do	[70] .530
Mine 6 [68]	Radial top-slicing	Gelatin dynamite, 35 percent		[70] .473
		Gelatin dynamite, 60 percent	Fuse and caps	.061
Mine 7 [68]	do	Gelatin dynamite, 35 percent	do	[70] .396
		Gelatin dynamite, 60 percent	do	.003
Mine 8 [68]	do	Special gelatin, 35, 40, 60 percent	do	[70] .547
Mine 9 [68]	Parallel top-slicing	Gelatin dynamite, 50 and 60 percent	do	[70] .372
Mine 10 [68]	Radial top-slicing	Ammonia gelatin, 50 and 60 percent	do	[70] .244
Mine 11 [68]	Radial and parallel top-slicing	do	do	[70] .482
Mine 12 [68]	Parallel top-slicing	Gelatin dynamite, 50 and 60 percent	do	[70] .319
Mine 13 [68]	do	do	do	[70] .353
Mine 14 [68]	do			[70] .467

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[54] D'Arcy, R. L., Mining Practice and Methods at the United Verde Extension Mining Co., Jerome, Ariz.: Bureau of Mines Inf. Circ. 6250, 1930, 11 pp.

[55] Vanderburg, W. O., Mining Methods and Costs at the Argonaut Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6311, 1930, 14 pp.

[56] Wade, J. W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Bureau of Mines Inf. Circ. 6360, 1930, 21 pp.

[57] Richard, F. A., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, N. Mex.: Bureau of Mines Inf. Circ. 6377, 1930, 13 pp.

[58] Spiers, James, Mining Methods and Costs at Central Eureka Mine, Amador County, Calif.: Bureau of Mines Inf. Circ. 6512, 1931, 12 pp.

[59] Skidmore, Joe H., Gold Mining and Milling Methods and Costs at the Gold Hill Mine of Talache Mines, Inc., Quartzburg, Idaho: Bureau of Mines Inf. Circ. 6985, 1938, 15 pp.

[60] Calhoun, A. B., Mining Methods at Bawdwin Mine: Trans. Am. Inst. Min. and Met. Eng. (issued with Min. and Met.), vol. 69, August 1923, pp. 208–247.

[61] Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Bureau of Mines Inf. Circ. 6107, 1929, 33 pp.

[62] Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Bureau of Mines Inf. Circ. 6167, 1929, 27 pp.

[63] Webb, J. S., and Skinner, T. W., Mining Methods and Costs at the Braden Copper Co.'s Mines, Sewell, Chile: Bureau of Mines Inf. Circ. 6565, 1932, 13 pp.

[64] Maclennan, F. W., Miami Copper Co., Method of Mining Low-Grade Ore Body: Trans. Am. Inst. Min. and Met. Eng., vol. 91, 1930 (Yearbook), pp. 39–86.

[65] Schaus, O. M., Method and Cost of Mining Hematite at the Eureka-Asteroid Mine on the Gogebic Range, Gogebic County, Mich.: Bureau of Mines Inf. Circ. 6348, 1930, 13 pp.

[66] Schaus, O. M., Mining Methods and Costs at the Montreal Mine, Montreal, Wis.: Bureau of Mines Inf. Circ. 6369, 1930, 29 pp.

[67] Bowen, R. A., Sublevel Caving, Large-Pillar Method, at the Montreal Mine: Am. Inst. Min. and Met. Eng., Mining Technology, Tech. Pub. 886, March 1938.

[68] Jackson, Chas. F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, 296 pp.

[69] Stoping and development, per long ton.

[70] Stoping only, per long ton.

[71] Haselton, W. D., Underground Mining Practice and Costs at a Mesabi Range Minn.) Mine Using the Top-Slicing System: Bureau of Mines Inf. Circ. 6325, 1930, (11 pp.

power scraping equipment, it is usually more economical to scrape the broken ore down on the footwall and thus avoid driving drifts and raises in the barren rock of the footwall (fig. 111). When the dip is only slightly less than the angle of repose of the broken ore, stationary steel slides alone or steel slides in which a little water is running are sometimes employed to convey the ore from the stope to the level. Where the dip is still flatter, steel shaking chutes have been used successfully. The installation of steel slides or shaking chutes consumes time and is costly; they are subject to damage from blasting or falls of ground and are a source of annoyance and expense. Where power is available, scrapers are much more flexible and efficient.

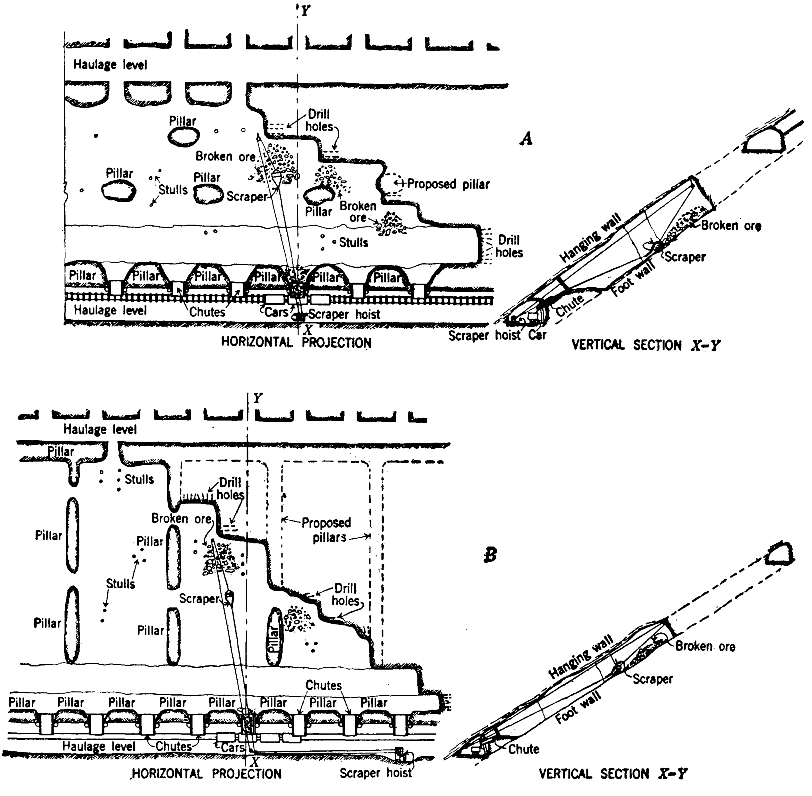
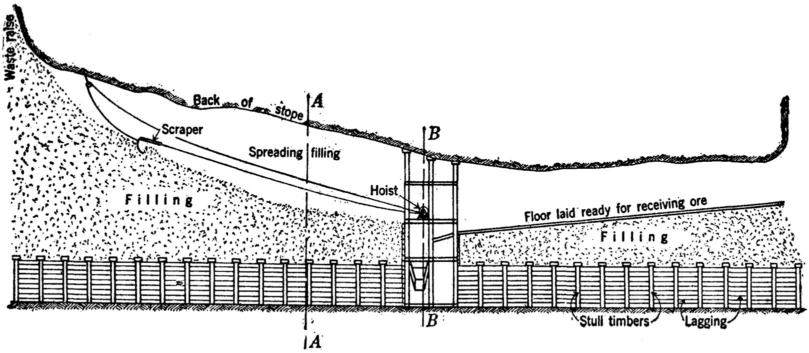

Y
Haulage level
Pillar
Drill holes
Broken ore
Pillar
Stulls
Pillar
Scraper
Proposed pillar
Pillar
Broken ore
Drill holes
Stulls
Pillar
Pillar
Pillar
Pillar
Pillar
Pillar
Haulage level
Chutes
Cars
Scraper hoist
X
HORIZONTAL PROJECTION
A
Hanging wall
Broken ore
Scraper
Foot wall
Chute
Scraper hoist
Car
VERTICAL SECTION X-Y
Y
Haulage level
Pillar
Stulls
Drill holes
Broken ore
Pillar
Pillar
Proposed pillars
Stulls
Scraper
Drill holes
Pillar
Pillar
Pillar
Chutes
Pillar
Pillar
Pillar
Pillar
Pillar
Pillar
Pillar
Pillar
Haulage level
Chutes
Cars
X
HORIZONTAL PROJECTION
Scraper hoist
B
Hanging wall
Broken ore
Scraper
Foot wall
Chute
VERTICAL SECTION X-Y

FIGURE 111.—Open stoping in Michigan copper mines. *A*, Large open stope with irregular pillars. *B*, Long, narrow stopes separated by long, narrow pillars.

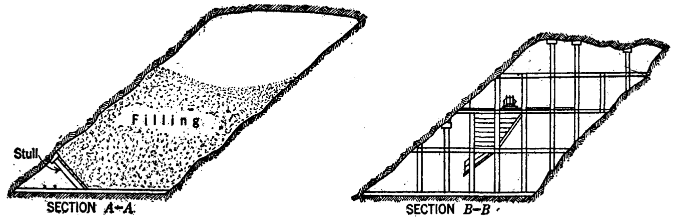
Scrapers are employed extensively for dragging broken ore to the ore passes in horizontal or slightly inclined cut-and-fill stopes (fig. 112). Figure 112 shows a scraper spreading filling at the left end of the stope. When the back has been blasted and broken down onto the floor at the right end of the stope the scraper is moved to that side without moving the hoist, and the ore is dragged into the chute. When all factors are considered, horizontal cut-and-fill stoping using scrapers or cars is often cheaper than gravity handling in rill stopes; reduced efficiency of the miners when working on an inclined floor and extra labor for bringing in and placing prop timbers and flooring and for setting up and rigging drills may more than offset the cost of scraping.

In square-set stopes, wing slides are used to divert broken ore from one tier of sets to another that is used as an ore pass. These are merely planks extending from one cap or girt to the next one diagonally below it. Their use considerably reduces the amount of hand shoveling, but to eliminate it entirely by this method when working large horizontal floors a large number of chutes is required. When working in vertical panels two to four sets wide, the use of wing slides will virtually eliminate hand shoveling in the stopes. Wing slides are also employed in stulled stopes (fig. 94).

Scrapers have been used successfully in square-set stopes working large horizontal floors. Thus, at the Park Utah mine[39] it is stated that scrapers have reduced stoping costs 30 percent. There the scraper hoist is mounted on a truck that is moved parallel to the hanging wall of the stope over a 3-inch plank floor to any desired line of sets.



VERTICAL LONGITUDINAL PROJECTION



SECTION A-A

SECTION B-B

FIGURE 112.—Scraping in cut-and-fill stope.

In flat beds or other flat deposits worked by breast stoping or heading-and-bench methods, the ore must be loaded from the floor into cars. Hand shoveling, scrapers, and power shovels are all employed for this purpose. Success with pan and belt conveyors for loading in rooms and entries in coal mines suggests their probable future use in metal mines.

At *A*, figure 113, is shown an electric shovel loading into cars in a heading-and-bench open stope, and at *B* scraper loading from an open breast stope into a string of cars on a main haulageway. Figure

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[39] Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Bureau of Mines Inf. Circ. 6290, 1930, p. 12.

114 shows an electric scraper hoist pulling ore into a mill hole in a large open stope.

In top slicing and sublevel caving, as practiced in the Lake Superior iron mines, scrapers are used exclusively for dragging the broken ore into raises (fig. 115 at *A*), or, where top slicing is done in thin ore bodies of low dip angle, for loading cars direct (fig. 115 at *B*). When loading directly into cars, a slide is constructed as shown. In thick deposits where haulageways can be driven mostly in ore under most of the slice area, with raises to the sublevels also in ore, the ore is usually scraped to raises, as shown at *A*.

Mechanical handling reaches maximum efficiency when there is enough broken ore to keep the equipment working at capacity or where quick removal of the ore greatly increases the output of the stope by permitting earlier resumption of ore breaking. One advantage of scraping not possessed by other methods is that scrapers often can be used to clean out ore in portions of stopes that it is unsafe to enter. Thus, a fall of ore sometimes may be cleaned out by a scraper and waste filling dragged in to take its place without endangering the lives of the miners.

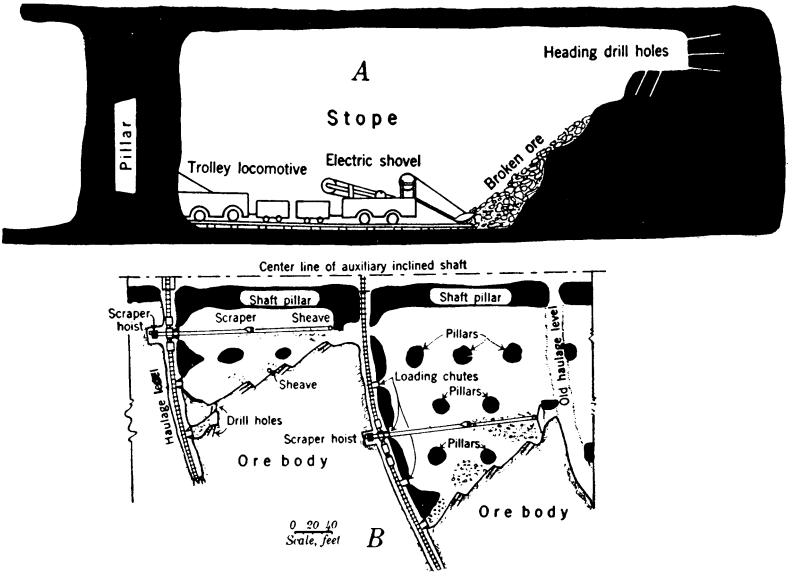

Heading drill holes
A
Stope
Pillar
Trolley locomotive
Electric shovel
Broken ore
Center line of auxiliary inclined shaft
Shaft pillar
Shaft pillar
Scraper hoist
Scraper
Sheave
Pillars
Old haulage level
Haulage level
Sheave
Loading chutes
Pillars
Drill holes
Scraper hoist
Pillars
Ore body
Ore body
0 20 40
Scale, feet
B

FIGURE 113.—Mechanical loading in flat open stopes. *A*, Electric shovel loading into cars on stope track. *B*, Power scraper loading in breast stope to cars on main haulageway.

Ore handling in stopes is intermittent when considerable sorting of ore and waste is necessary. Obviously, mechanical methods are greatly handicapped under this condition and therefore hand shoveling usually is preferred. Likewise, if only a small tonnage is to be handled, the expense of installing mechanical equipment is not warranted.

Hand shoveling directly into the ore passes or into wheelbarrows is practiced where the passes are spaced at short intervals. Wider spacing may make it more advantageous to lay track and use cars for transferring the broken ore from the muck pile (fig. 93). Cars are used more generally in the wider stopes and direct shoveling and

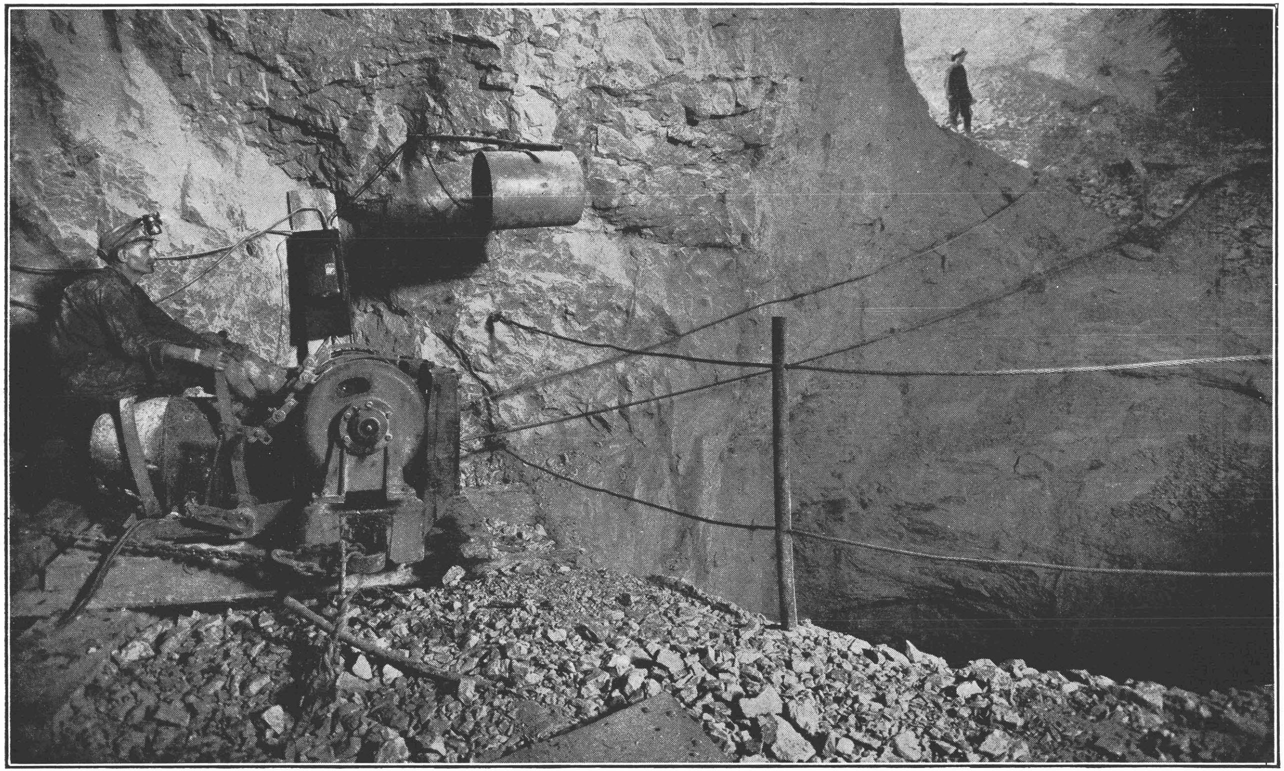
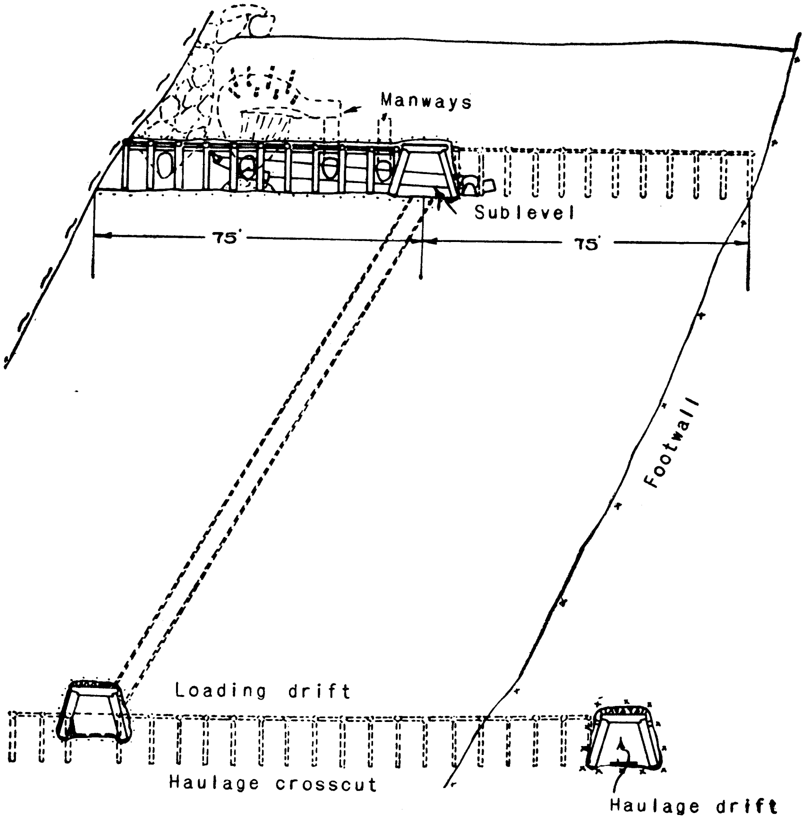


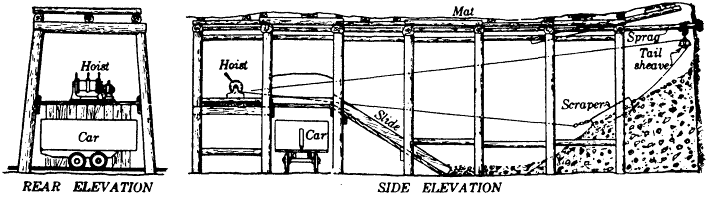
FIGURE 114.—Scraper dragging ore from stope face into mill hole, Mascot, Tenn.

wheelbarrows in narrow stopes. In wide stopes scraping often will prove more economical than cars unless considerable sorting is necessary, and, hence, the choice of methods will be influenced largely by the requirements for sorting.


Manways
Sublevel
75'
75'
Footwall
Loading drift
Haulage crosscut
Haulage drift

VERTICAL CROSS-SECTION

A


Hoist
Car
REAR ELEVATION
Mat
Sprag
Tail sheave
Hoist
Scrapers
Slide
Car
SIDE ELEVATION

B

FIGURE 115.—Power scraping in sublevel caving and top-slicing. *A*, Scraping to raises (sublevel caving) *B*, Scraping to cars (top-slicing).

FILLING

Stope filling has been discussed by Jackson [40] in an earlier paper. Filling long has been recognized as the most effective method of permanently supporting the walls of mine workings and minimizing movement and subsidence. It is used alone in cut-and-fill stoping and in conjunction with framed timbers in square-set stoping for support of weak walls during the working of the stope and for permanent support. In cut-and-fill stoping its function is to support the walls and to provide a working floor; loose filling cannot be packed tight enough against the back to hold great downward pressure and under subsidence of the back it is compressible. Dry-pack walls have been employed widely, especially in Europe, both to afford direct support to the back and to retain loose filling.

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[40] Jackson, Chas. F., Waste Filling of Stopes: Bureau of Mines Inf. Circ. 6816, 1935, 24 pp.

Stope filling may be introduced (1) at frequent regular intervals during stoping as one of the operations in a cycle, as in cut-and-fill stoping, (2) after completion of a stope to afford permanent support, or (3) to fill stopes in preparation for mining adjacent blocks of unmined ore, such as pillars between stopes and crown or floor pillars. Filling also is employed in emptied shrinkage stopes for the dual purpose of providing permanent support to the walls and a rising floor from which tongues of ore in the walls can be reached and the walls can be prospected for blind lodes at various horizons. Mill sands have been employed at Franklin, N. J., to fill the space above top-slice caves in pillar mining between filled stopes and to provide a cushion above the top slices to absorb the shock of any masses of rock capping that may break loose. Waste filling is used for a similar purpose in the Colorada ore body at Cananea, Mexico. Filling also may be employed under some special conditions for extinguishing mine fires.

Various filling materials are employed, such as waste rock from development or from sorting in the stopes, waste shot off the walls of the stope or broken in waste raises and crosscuts driven into the walls, rock from surface quarries or sand and gravel pits, mill-tailing sands, and granulated or crushed slag.

Rock from development often can be disposed of most cheaply by using it for stope filling and may make it unnecessary to break waste, especially for filling. In other instances the supply of development rock may be inadequate and have to be supplemented by filling material from other sources.

Where movement and subsidence over and around the workings must be prevented, a filling that will settle or shrink the least after placement is required. Sand filling usually shrinks the least of any filling material in common use, whereas coarse rock will shrink considerably by reason of the gradual filling of the voids. Eaton [41] states that coarse waste may compress as much as 10 to 25 percent before the fill develops its full strength. At the Tintic Standard mine [42] filling in square-set stopes settles as much as 7 feet between levels 120 feet apart.

Coarse filling is objectionable in cut-and-fill stoping, as it increases the difficulty of laying an even floor under the next cut of ore.

The introduction and spreading of fill in the stopes constitute important operations in cut-and-fill and square-set stopes. In rill stoping, gravity is employed for this purpose, and hand work usually is required only for spreading the fill immediately under the hanging wall and for filling corners.

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[41] Eaton, Lucien, Sand Filling Through Pipes and Boreholes: Trans. Am. Inst. Min. and Met. Eng., vol. 102, 1932, pp. 33–42.

[42] Wade, James W., Mining Methods and Costs at the Tintic Standard Mine, Tintic District, Utah: Bureau of Mines Inf. Circ. 6360, 1930, p. 11.

In flat stopes, filling may be placed by hand, by power scrapers (fig. 112), or by wheelbarrows or cars (fig. 93). Sand is the easiest filling material to spread, especially if scraping is employed, and, in addition to conventional methods of handling, blowing or flushing may be used. Thus, at Copper Range, stamp sands were at one time blown by compressed air from a hopper through a 4-inch pipe for distances up to about 250 feet and for 20 feet beyond the end of the pipe; air pressure was 75 to 80 pounds.[43] By blowing, sand can be placed above the level of the intake and under bad ground where it is unsafe for men to work.

At Matahambre, Cuba,[44] about half the fill tonnage was made up of mill sands in 1929. Mill tailings are pumped to a 30-foot bowl classifier on surface near a raise from the mine workings. The sand or rake product, to which water has been added, is taken through a special rubber-lined pipe down the raise and into the stopes to be filled. Classified sands are also employed for filling in some of the mines on the East Rand in Africa.[45] The sand from the classifier falls into a hopper above a borehole, and enough water—1 part to 2 or even 3 parts of sand—is added to make it flow down the hole (6 inches or more in diameter). Underground the sand-water mixture is distributed in launders set at 5- to 30-percent gradient. The sand packs solidly and provides almost perfect support for the hanging wall. Costs are given as $0.185 per cubic yard of sand placed.

Wet sand that drains quickly and perfectly forms a fill that will shrink very little. One objection to the use of water, of course, is the cost of handling it as it drains from the fill. Another is the difficulty in sealing off the fill so that the water will drain immediately but without carrying sand with it.

Costs of filling at different mines are not available on a comparable basis; some reports give the cost of procurement of filling but not the cost of placement, whereas a number of others give only the cost of placing; some give the cost of filling per ton of ore mined and others the cost per yard of filling. At the Tintic Standard mine, waste was broken in an open-cut and delivered to the shaft at a contract price of 20 cents per ton. In one open-cut at Cananea, 9 to 10 men delivered 310 tons of waste rock per day at a cost of 13 cents per ton. The filling cost at Cananea during August and September 1929, in square-set, horizontal cut-and-fill, and rill cut-and-fill stopes, averaged 17 cents per ton of ore mined; in 1937 the cost of filling over top slices between completed cut-and-fill stopes was $0.1704 per ton mined by top-slicing.

In 1928 the cost of filling at Matahambre, including both rock and sand fill, was $0.177 per ton mined.

In 1929 the labor cost of filling in cut-and-fill stopes at the Pecos mine was 25 cents per ton of ore mined, in square-set stopes 18 cents, and in underhand pillar stopes 26 cents.

At the Hollinger mine, Timmins, Ontario, sand filling is excavated from a pit and transported to the mine over an aerial tramway 3.6

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[43] Schacht, W. H., Mining Methods of the Copper Range Co.: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 346–370.

[44] Richert, George L., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

————, Filling Stopes with Mill Tailing: Eng. and Min. Jour., vol. 127, No. 19, Mar. 2, 1929, p. 348.

Homer, D. D., Rubber Pipe Lining Minimizes Pulp Abrasion: Eng. and Min. Jour., vol. 132, No. 8, Oct. 26, 1931, pp. 367–368.

[45] Eaton, Lucien, Work cited in footnote 41.

miles in length. In 1934, the cost of excavating and transportation was $0.1494 per ton of sand, including operation, power, maintenance, and amortization of cable wear.[46] The cost of filling in the stopes (horizontal cut-and-fill, fig. 93) during the same year averaged $0.1221 per ton of ore broken.[47]

At the McIntyre mine, Schumacher, Ontario, in 1930, it cost 50 cents per ton to break waste in stope crosscuts and 50 cents per ton to handle it in cars and spread it by hand. In a large waste stope, waste was broken by sublevel stoping at a cost of 10 cents per ton; secondary breaking on grizzlies, loading, and tramming to the main waste pass cost 25 cents per ton; and waste-stope development, grizzly installation, etc., about 10 cents per ton, a total of 45 cents per ton for waste procurement.

At Matahambre the cost of procuring, handling, and distributing mill-sand filling was as follows:

	Cost per ton of sand
Labor	$0. 105
Burlap (for ceiling off ore passes and manways through the fill)	. 048
Pipe	[1] . 071
Pumping	. 006
Power	. 020
	. 250

[1] Pipe not worn out.

Actual cost, allowing for remaining useful life of pipe, was probably not over 20 cents per ton.

TIMBERING

Timbering in stopes ranges from occasional light props or stulls to temporarily support patches of unsafe ground to regular timbering with square sets and large bulkheads or "cribs" of solid timbers.

Regular stull timbering has been mentioned. Figure 94 shows this type of timbering in a narrow, steep-dipping vein. Figure 116 shows regular stull timbering in the Conglomerate mine, Calumet, Mich., where mining operations were conducted at great depths without filling [48] in a flat-dipping lode. A retreat system of mining was employed, whereby stopes 100 feet long were worked up quickly from one level to the next, using clusters of heavy stulls for temporary support and then allowing the hanging wall to crush down on the timbers and cave. Standard practice in ore up to 20 feet thick was to use four rows of stulls in clusters of two 18-inch to 24-inch-diameter timbers alongside each other, set 7 feet apart between clusters, and in rows on 7-foot centers (fig. 116). Above the four rows, single stulls were used since the upper part of the stope did not have to remain open very long. A double row of "breaker" stulls was set along the end of each 100-foot section to confine the breakage of the hanging wall to the worked out section inby of the section being worked. The heavy timbers used in this mine had to be lifted into position by block and tackle.

Stulls are used incidentally in shrinkage stopes and either incidentally or in a systematic manner in cut-and-fill stopes, as shown in

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[46] Williams, Maurice E., Back-Fill.—Procurement, Transportation, and Distribution: Canadian Min. Jour., September 1935, pp. 418–420.

[47] Young, Arthur W., Mining: Canadian Min. Jour., September 1935, p. 387.

[48] Vivian, Harry, Deep Mining Methods, Conglomerate Mine of the Calumet & Hecla Consolidated Copper Co.: Bureau of Mines, Inf. Circ. 6526, 1931, 20 pp.

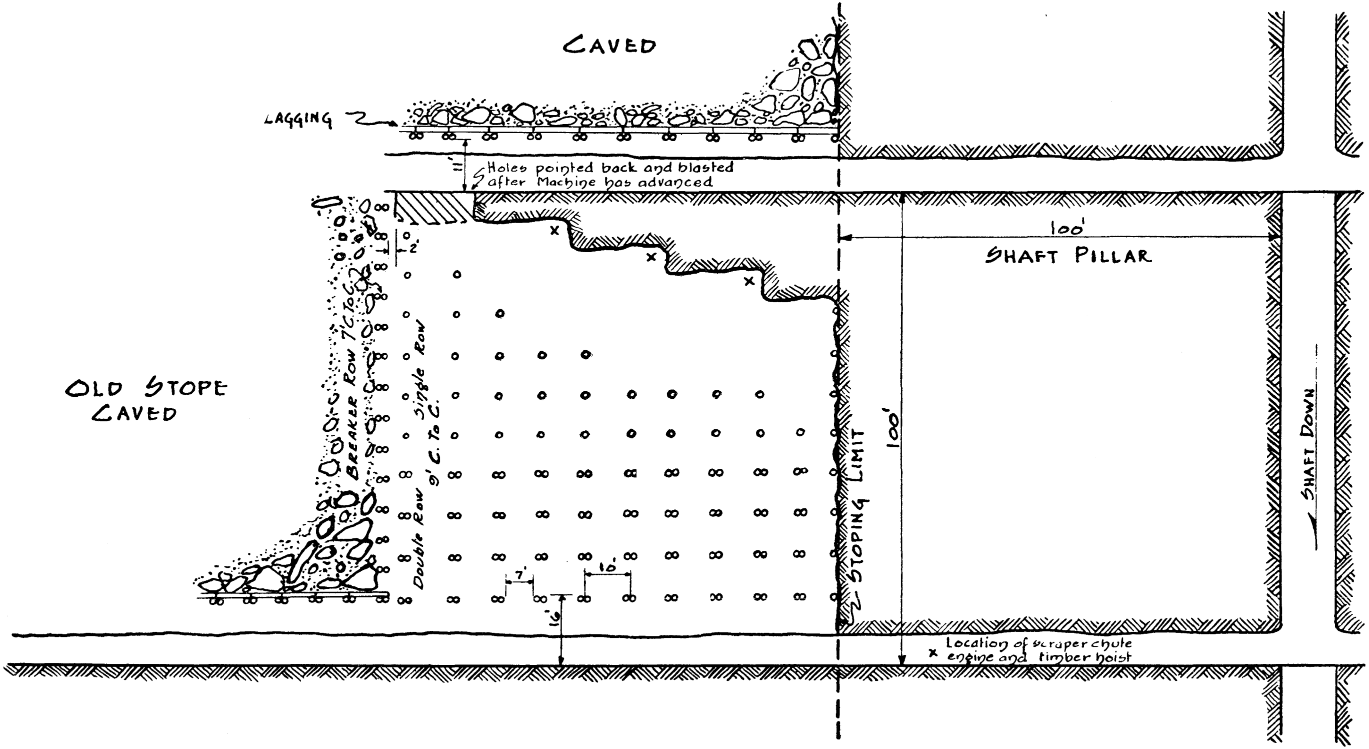
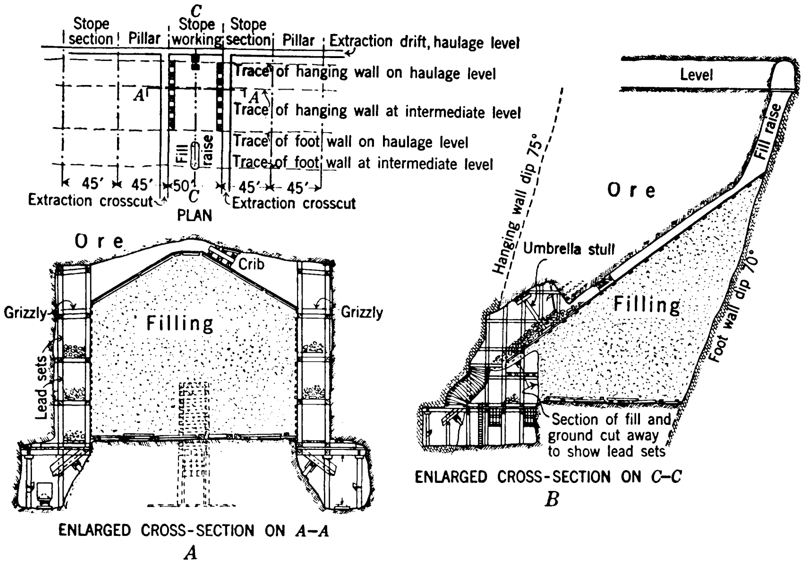

CAVED
LAGGING
11'
Holes pointed back and blasted after Machine has advanced
OLD STOPE CAVED
2'
BREAKER ROW 7' C. TO C.
Single Row 9' C. To C.
Double Row
7'
10'
16'
100'
SHAFT PILLAR
100'
STOPING LIMIT
SHAFT DOWN
x Location of scraper chute engine and timber hoist

FIGURE 116.—Stull stoping, Conglomerate lode, Michigan.

figure 92 (stringer-set-and-fill system). In the latter instance vertical posts generally are set between the stulls, and girts or ties are used for horizontal bracing. To support weak spots in the back, occasional props are often set on the fill, with a headboard on top and under the stope back and with sill boards underneath on top of the fill. In wide stopes with a slabby back, temporary stringers supported on posts may be employed, as shown in figure 93, in which case most of the timber may be recovered and used again. Larger areas of heavy back may be supported on cribs as shown in figure 117, which shows inclined cut-and-fill stoping between lines of square-set timbers at the Campbell mine, Bisbee, Ariz. At *B* in this figure is also shown an umbrella stull with several laggings laid across the headboard to cover more hanging-wall area.


C
Stope section
Pillar
Stope working
Stope section
Pillar
Extraction drift, haulage level
Trace of hanging wall on haulage level
A
A
Trace of hanging wall at intermediate level
Trace of foot wall on haulage level
Trace of foot wall at intermediate level
Fill raise
45′
45′
50′
45′
45′
Extraction crosscut
C
Extraction crosscut
PLAN
Ore
Crib
Grizzly
Grizzly
Filling
Lead sets
Level
Fill raise
Ore
Hanging wall dip 75°
Umbrella stull
Filling
Foot wall dip 70°
Section of fill and ground cut away to show lead sets

ENLARGED CROSS-SECTION ON *A–A*
A

ENLARGED CROSS-SECTION ON *C–C*
B

FIGURE 117.—Inclined cut-and-fill stoping between lines of square-sets, Campbell mine, Bisbee, Ariz.

In square-set stoping, the timbers may be either round or sawed but in either case are framed at the ends to fit together so that, as pressure develops, the joints tighten (fig. 95, *A*). The framing at different mines varies considerably in detail, but there are two general types of joints—the post-butting and the cap-butting type. Figure 118 shows several typical methods of framing. At *A* and *B* are shown, respectively, sawed timbers framed with post- and cap-butting joints; at *C* and *D* are shown methods of framing round timbers; and *E* illustrates the Moore diagonal set, in which the posts are replaced by diagonal members, a useful method where, for any reason, a vertical post cannot be set. Gardner and Vanderburg [49] have compared the post-butting and cap-butting types as follows:

> Although the post-butting type has an advantage under some conditions and continues to be used, present practice tends toward the adoption of the cap-butting type of framing. Where both side and top pressures are encountered the cap-

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[49] Gardner, E. D., and Vanderburg, W. O., Square-Set System of Mining: Bureau of Mines Inf. Circ. 6691, 1933, pp. 12–16.

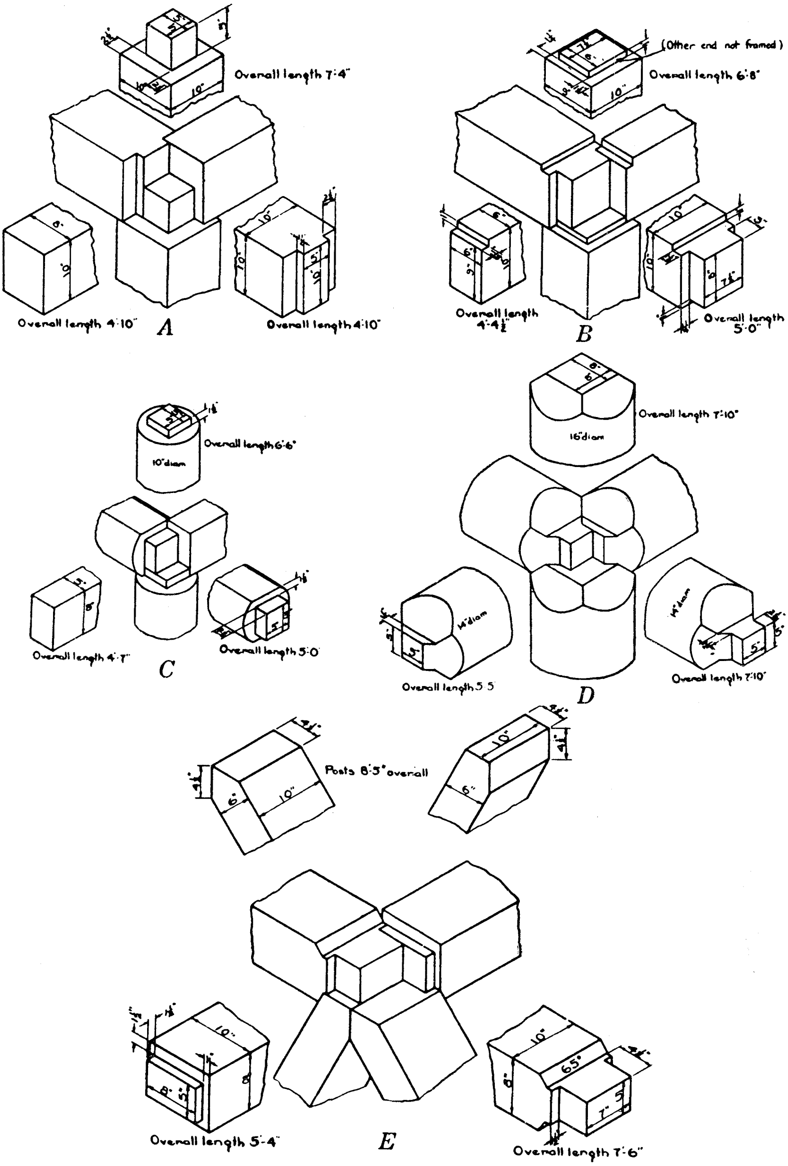
Overall length 7'-4"
Overall length 4'-10"
A
Overall length 4'-10"
(Other end not framed)
Overall length 6'-8"
Overall length 4'-4½"
B
Overall length 5'-0"
Overall length 6'-6"
10" diam
Overall length 4'-7"
C
Overall length 5'-0"
Overall length 7'-10"
16" diam
14" diam
14" diam
Overall length 5'-5"
D
Overall length 7'-10"
Posts 8'-5" overall
Overall length 5'-4"
E
Overall length 7'-6"

FIGURE 118.—Typical methods of framing square-set timbers. *A*, Cananea (Mexico) post-butting joint. *B*, Bingham (Utah) cap-butting joint. *C*, Page (Idaho) cap-butting joint, round timbers. *D*, Beveled joint on round timbers. *E*, Moore diagonal set.

butting type is preferable, as the long horn on a post of a post-butting set is a point of weakness. The strength of a post is proportional to the area of cross section over which the load is distributed. If a post has an area of 12 by 12 inches and the bearing on its end is confined to a 6- by 6-inch area, the timber will begin to fail as soon by crushing as though its full cross section were only 6 by 6 inches; consequently, the post-butting joint is not as strong as the other. The pressure coming on the horn of a post has a tendency to split and spread it and in so doing may force the caps and girts from their seats on the post. Furthermore, the bearing area of the horn is small laterally, and if the horn is crushed when pressure comes upon it the sets may be forced out of alinement. These objections are largely eliminated with the cap-butting joint, because where the caps butt against each other the horn or tenon on the post is short and has a relatively large cross-sectional area; consequently, when the caps are compressed the weight is carried over the greatest cross section of the post instead of on the long, narrow horn. A joint with the post-butting also generally has the disadvantage of a relatively deep mortise at the top; this mortise often becomes filled with fine ore from blasting, so that if it is not entirely cleaned out an imperfect joint results when the next post above is placed. Moreover, miners are disposed to cut off the horn at the bottom of the post instead of cleaning out the hole to fit the horn.

In framing square-set joints a relatively larger proportion of the original timber is cut away in making the horns for a post-butting set than in forming the tenons on a cap-butting set. Consequently a little more timber is required for a set of the former type. However, as shown in table 35, fewer cuts with the saw are required in framing a typical post-butting set; this extra framing for a cap-butting set about offsets the saving in timber over the other type of set.

(Table 35 also gives other details covering types and sizes of square-set timbering employed at a number of individual mines.)

In the post-butting type of framing, the horns usually are the same length on each end of the post and both ends are framed symmetrically.

TABLE 35.—*Types and sizes of square sets*

Figure	Name of mine or set	Location	Type of framing	Section, inches (rd=round)			Over-all lengths of members			Center-to-center dimensions			Saw cuts, per set [1]	Ore per set, cubic feet [2]	Timber per set,[3] board feet	Timber per cubic foot of ore, board feet
				Post	Cap	Girt	Post	Cap	Girt	Vertical	Capwise	Girtwise				
							Ft. in.	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*	*Ft. in.*				
2, *A*	Cananea set	Sonora, Mexico	Post-butting	10 x 10	10 x 10	8 x 10	7 4	4 10	4 10	7 4	5 3	5 3	24	202	. 133	0. 66
	Goldfield set	Goldfield, Nev	do	8 x 8	8 x 8	6 x 8	7 4	4 7	4 7	7 4	5 0	5 0	24	183	82	. 45
	Tintic Standard mine (old type).	Dividend, Utah	do	8 x 8	8 x 8	8 x 8	7 0	5 8	5 8	7 0	6 0	6 0	24	252	98	. 39
	Homestake mine	Lead, S. Dak	do	12 x 12	12 x 12	11 x 12	8 3	5 6	5 6	8 3	6 0	6 0	40	297	226	. 76
	Bunker Hill and Sullivan (square posts).	Kellogg, Idaho	do	10 x 10	8 x 10	8 x 10	7 10	4 8	4 8	7 10	5 0	5 0	24	196	127	. 65
2, *B*	Bingham set	Bingham, Utah	Cap-butting	9 x 10	10 x 10	6 x 10	6 8	5 0	4 4	7 4	5 0	5 0	36	183	114	. 62
3, *A*	Portland post-butting set.	Cripple Creek, Colo	Post-butting	10 x 10	10 x 10	10 x 10	7 3	5 4	5 4	7 3	5 8	5 8	40	233	149	. 64
3, *B*	Burlingame set	Butte, Mont	do	10 x 10	10 x 10	10 x 10	7 0	5 0	4 6	7 0	5 6	5 0	40	193	138	. 71
	Vulcan mine	Vulcan, Mich	do	12 x 12	12 x 12	12 x 12	7 10	7 0	6 7	8 0	7 5	7 0	24	416	257	. 62
	Bunker Hill and Sullivan (round posts).	Kellogg, Idaho	do	10–14rd	8 x 10	8 x 10	6 10 or 7 10	4 8	4 8 or 9 8	6 10	5 0	5 0	24	171	127	. 74
3, *C*	Pecos mine	Tererro, N. Mex	do	10–12rd	11–13rd	8–10rd	6 7	4 8	4 8	6 7	5 0	5 0	48	165	121	. 73
	Coeur d'Alene set	Wallace, Idaho	do	10–12rd	8 x 10	8 x 10	8 0 or 7 10	4 8	4 8 or 9 8	8 0	5 0	5 0	24	200	126	. 63
3, *D*	Center Star mine	Rossland, British Columbia.	do	12–15rd	12–15rd	12–15rd	8 0	4 10	4 10	8 0	5 4	5 4	56	227	210	. 93
4, *A*	Butte set	Butte, Mont	Cap-butting	10 x 10	10 x 10	6 x 10	7 0	5 0	4 6	7 6	5 0	5 0	32	188	123	. 65
	do	do	do	12 x 12	12 x 12	8 x 12	7 0	5 0	4 4	7 8	5 0	5 0	32	192	179	. 93
	Copper Queen mine	Bisbee, Ariz	do	10 x 10	10 x 10	5 x 10	7 5	5 0	4 7	7 10	5 0	5 0	32	196	123	. 63
	Aspen set	Aspen, Colo	do	10 x 10	10 x 10	10 x 10	6 6	5 4	4 10	7 0	5 4	5 4	40	199	139	. 70
4, *B*	Ruth mine	Ruth, Nev	do	10 x 10	10 x 10	10 x 10	7 2	5 6	5 2	7 8	5 6	5 8	40	239	149	. 62
4, *C*	Esperanza set	Esperanza, Mexico	do	8 x 8	8 x 8	8 x 8	7 6	5 0	4 6	8 0	5 0	5 0	24	200	91	. 45
4, *D*	Richmond mine	Eureka, Nev	do	12 x 12	12 x 12	10 x 10	6 2	5 0	4 0	6 10	5 0	4 9	48	161	168	1. 04
	Calumet and Arizona mine.	Warren, Ariz	do	10 x 10	10 x 10	10 x 10	7 4	5 0	4 6	7 10	5 0	5 0	40	196	140	. 71
4, *E*	Tintic Standard mine	Dividend, Utah	do	9 x 10	10 x 10	6 x 10	6 4	6 0	5 4	7 0	6 0	6 0	28	252	124	. 49
	Park Utah mine	Keetley, Utah	do	9 x 10	10 x 10	6 x 10	6 8	5 0	4 4	7 4	5 0	5 0	28	183	113	. 62
	Magma mine	Superior, Ariz	do	10 x 10	10 x 10	8 x 10	6 0	5 0	4 6	6 8	5 0	5 0	40	167	122	. 73
	United Verde mine	Jerome, Ariz	do	10 x 10	10 x 10	10 x 10	6 7	5 6	4 11	7 2	5 6	5 6	40	217	142	. 65

[1] Right-angle cuts are counted as 2 cuts, beveled cuts as 1. Slabbing cuts are not counted. Where sets are framed underground, framing cuts are not counted. A set consists of 1 post, 1 cap, and 1 girt.

[2] Based on center-to-center dimensions of sets.

[3] The board-foot content of round timbers is calculated on the basis of the total volume of the timbers. The formula used is: Bd. ft.$=0.06545\ d^{2}1$, in which the diameter (d) is expressed in inches, and the length (1) in feet.

TABLE 35.—*Types and sizes of square sets*—Continued

Figure	Name of mine or set	Location	Type of framing	Section, inches (rd.=round)			Over-all lengths of members						Center-to-center dimensions						Saw cuts, per set	Ore per set, cubic feet	Timber per set, board feet	Timber per cubic foot of ore, board feet
				Post	Cap	Girt	Post		Cap		Girt		Vertical		Capwise		Girtwise					
							Ft.	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*	*Ft.*	*in.*				
4, *E*	United Verde Extension mine (2 sets).	Jerome, Ariz	Cap-butting	10 x 10	10 x 10	10 x 10	7	7	5	6	4	11	8	2	5	6	5	6	40	247	150	0.61
				8 x 8	8 x 8	8 x 8	7	9	5	6	5	1	8	2	5	6	5	6	40	247	98	.40
	Leadville set	Leadville, Colo	do	10 x 10	10 x 12	7 x 12	6	3	6	10	5	3	7	0	6	10	5	10	36	279	157	.56
	Tonopah Extension mine.	Tonopah, Nev	do	10 x 10	10 x 10	10 x 10	7	6	6	0	6	0	8	1	6	0	6	7	40	319	163	.51
	Eureka set	Eureka, Nev	do	12 x 12	12 x 12	12 x 12	6	6	5	8	4	4	7	0	5	8	5	0	40	194	196	1.01
	Portland cap-butting set.	Cripple Creek, Colo	do	8 x 8	8 x 8	8 x 8	6	6	5	0	4	6	7	0	5	0	4	11	44	172	85	.49
4, *F*	New Idria mine	New Idria, Calif	do	10 x 10	8 x 8	8 x 8	6	6	5	0	4	6	6	11	5	0	4	10	48	167	105	.63
	Pilares mine	Sonora, Mexico	do	12rd	8 x 10	6 x 10	7	6	5	0	4	6	8	0	5	0	5	0	32	200	127	.64
	Arizona Copper Co. mine.	Morenci, Ariz	do	12rd	8 x 8	8 x 8	6	9	5	0	4	7	7	2	5	0	5	0	40	179	115	.64
5, *A*	Page mine	Kellogg, Idaho	do	10rd	10rd	5 x 8	6	6	5	0	4	7	6	11	5	0	5	0	32	173	91	.53
5, *B*	Argonaut mine	Jackson, Calif	do	14–18rd	12 x 12	8–12rd	7	0	5	4	4	0	7	10	5	4	5	0	12	209	208	1.00
	Rossland set	Rossland, British Columbia.	do	12–20rd	12–16rd	12–16rd	8	2	5	4	4	4	8	10	5	4	5	0	44	235	261	1.11
	Clark mine	Butte, Mont	do	10–14rd	10 x 10	7–8rd	6	7	5	7	4	5	7	0	5	7	4	10	40	189	127	.67
5, *C*	Black Rock mine	do	do	10–12rd	10–12rd	10–12rd	7	7	5	10	5	5	8	0	5	10	5	10	40	272	149	.55
5, *D*	Anaconda Rocker set	do	do	14rd	14rd	12rd	7	5	5	4	5	0	7	9	5	4	5	4	[4] 112	220	211	.96
	Bawdwin mine	Burma	do	10rd or larger	10rd or larger	8rd or larger	7	0	5	6	5	2	7	4	5	6	5	6	80	222	103	.46
5, *E*	Brunton bevel set	Butte, Mont	do	16rd	14rd	14rd	7	10	7	10	5	5	8	3	7	10	5	10	36	377	301	.80
	Brunton square and bevel set.	do	do	16rd	16rd	16rd	7	10	5	10	5	10	8	3	5	10	6	3	64	301	327	1.09
5, *F*	Chillagoe mine	Queensland, Australia.	Cap-and-girt butting.	12rd	10rd	10rd	6	0	6	2	6	2	6	2	6	2	6	2	40	235	137	.58
	Unwarra V-framed set	Candor, N. C	Cap-butting	10rd	10rd	10rd	7	2	6	0	6	0	7	7	6	0	6	0	56	273	125	.46
6	Moore diagonal set	Tonopah, Nev	do	6 x 10	8 x 10	8 x 10 or 6 x 10	8	5	7	6	5	4	8	0	7	6	5	11	40	355	170	.48
7, *A*	Silver King stringer set.	Park City, Utah	do	8–10rd	8 x 8	6–8rd	6	6	10	0	4	4	6	10	10	0	5	0	None	342	[5] 150	.44
	Silver Hill stringer set	do	do	8 x 8 or 8–10rd	8 x 8	4 x 8 or 6–8rd	6	6	10	0	4	4	6	10	10	0	5	0	16	342	[5] 146	.43
7, *B*	Morning mine	Mullan, Idaho	do	12–16rd	12–18rd	10–12rd	8	0	8–16		4	0	8	7	8–16		5	0	4	510	[5] 445	.87
	Hecla and Star mines	Burke, Idaho	do	12–16rd	12–18rd	3 x 10 or 6 x 10	8	0	8–16		4 or 9		9	0	8–16		5 or 10		None	540 or 1,080	[5] 402 or 472	.75 or .44

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[4] More or less, depending on size of timber.

[5] In stull or stringer sets 5 timbers to a set; that is, 2 posts, 1 cap, and 2 girts.

Slice timber used in top slicing and sublevel caving is required to stand only a short time and is then blasted out or allowed to crush. Therefore, timber that would not be suitable for drift sets or square-set timbers may be acceptable for slicing. Slice timber, however, should be tough and strong, yielding or crushing gradually without breaking short, and green timber is, therefore, best. Specifications of the Cleveland-Cliffs Iron Co. some 15 years ago were as follows:[50]

Timber Specifications, Cleveland-Cliffs Iron Co.

Stull timber (posts and caps).—Hemlock, hard maple, soft maple, yellow birch, tamarack, and Norway pine. Must be sound, straight, and green. Ash, white birch, poplar, and balm of gilead not accepted. Tamarack in top diameters of 8 and 9 inches containing not more than ¼ inch sap rot in depth accepted; in diameters of 10, 11, 12, and 13 inches containing not more than ½ inch sap rot in depth. Lengths 8 and 16 feet.

Lagging.—Straight sound cedar, tamarack, and spruce (10 percent jack pine permitted). Round, 3 to 4½-inch top, larger than 4½ inches to be split. Split, not less than 2½ by 4 nor greater than 3 by 6 inches. In 5-foot lengths, 160 cubic feet per cord. In 7- and 8-foot lengths, 128 cubic feet per cord, not less than 125 pieces.

Covering-down boards.—No. 3 maple and birch. To be resawed from 2-inch maple and birch hearts to ⅝ inch. To be sound green lumber. Widths 6 inches and wider. Lengths not under 6 feet and not over 9 feet.

Since these specifications were written, exhaustion of some kinds of green timber has made it necessary to use varying proportions of dry timber. Tamarack and pine have become relatively scarce in the region.

Specifications for slice timber at three mines on the Mesabi range in 1930 were as follows:

Mine	Length of caps, feet	Diameter of cap timber, inches	Height of standard posts, feet	Diameter of post timber, inches
				Tops
1	8 or 10	10–12	Up to 18	7–9
2	10	7–9	11–12	7–9
3	8 to 12	8–10	10	8

At one mine on the Cuyuna range, specifications were as follows: Posts, 9- to 12-inch tops; caps, 12- to 15-inch tops; lagging, 5- to 7-inch tops; poles (for flooring), 3- to 5-inch tops.

Flooring may consist of mining boards (a grade below the lowest standard commercial grade) of 1-inch pine or ⅜- to ⅝-inch hardwood boards, round poles and boards, or round poles covered with wire fencing. Where heavy scrapers are used in the slices, the bottom may be covered with long poles or plank laid lengthwise. Wire fencing is used extensively instead of boards for covering the floor and sides of slices on the Lake Superior iron ranges, and the material cost is about the same as for boards per square foot covered but the cost of handling and laying is only about a quarter as much.

Slice timbers are simply drift sets either with almost vertical posts for use under shallow depths and light gob, or with a batter for use where the gob is heavy and there is considerable side pressure as well as top pressure; in sublevel caving, posts generally have a decided batter.

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[50] Elliott, S. R., Jopling, J. E., Chenneour, R. J., and Derby, E. L., Mining Methods of Marquette District, Michigan: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 122–138.

Data on timber consumption in stopes per ton of ore mined are given in table 36.

TABLE 36.—*Timber consumption in stopes* [1]

I. OPEN STOPING

Mine	Variation of stoping method	Timber consumption per ton of ore mined		Remarks
		Board feet [2]	Linear feet, round timber	
Marquette No. 1, Mich.[3]	Regular pillars	0.224		
Menominee No. 1, Mich.[4]	Sublevel stoping	.023		Soft ore.
Granada, Quebec [5]	Overhand in 45°-55° veins	1.650		Timber for stulls and wing slides.
Mt. Isa, Queensland [6]	Sublevel stoping	1.06		
II. SHRINKAGE STOPING				
Nevada-Massachusetts, Nev.[7]	On drift pillars	2.90	0.49	
Teck-Hughes, Ontario [8]	do	3.87		
Consolidated Cortez, Nev.[9]	do	1.36		
Verde Central, Ariz.[10]	On drift timbers	3.08		
Porcupine United, Ontario [11]	do	2.666		
Kirkland Lake Gold, Ontario [12]	do	1.750		
Eustis, Quebec [13]	On drift pillars	[14].376		
Vipond, Ontario [15]	Some on timbers; some on pillars.	.420	.70	
Elkoro, Nev.[16]	On drift timbers	1.355	.852	
Hog Mountain, Ala.[17]	Mostly on drift pillars	1.200		

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[1] Timber used in development not included.

[2] Sawed timber or round timber calculated to board feet. The figures given probably include chute and raise timbering in most instances.

[3] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[4] Eaton, Lucien, Mining Soft Ore by Open Stopes at Mine No. 1, Menominee Range, Mich.: Bureau of Mines Inf. Circ. 6180, 1929, 10 figs.

[5] Loofbourow, R. F., Mining Methods and Costs at Granada Gold Mines, Ltd., Rouyn, Quebec: Bureau of Mines Inf. Circ. 6709, 1933, 15 pp.

[6] Kruttschnitt, J., and Mann, V. I., Mining Methods and Costs at the Mount Isa Mines, Ltd., Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

[7] Heizer, O. F., Method and Cost of Mining Tungsten Ore at the Nevada-Massachusetts Mines at Mill City, Nev.: Bureau of Mines Inf. Circ. 6284, 1930, 13 pp.

[8] Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6322, 1930, 21 pp.

[9] Hezzelwood, G. W., Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.: Bureau of Mines Inf. Circ. 6327, 1930, 15 pp.

[10] Dickson, R. H., Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.: Bureau of Mines Inf. Circ. 6464, 1931, 13 pp.

[11] Tolman, J. D., Mining Methods and Costs at the Porcupine United Gold Mines' Rochester Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6470, 1931, 6 pp.

[12] Dumbrille, J. C., Mining Methods at Kirkland Lake Gold Mining Co. (Ltd.), at Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6490, 1931, 12 pp.

[13] Snow, Fred W., and Brownbill, H. F., Mining Methods and Costs at the Eustis Mine: Canadian Min. and Met. Bull. 286, February 1936, Transactions Sec., pp. 70–85.

[14] Mining and maintenance.

[15] Dye, R. E., Mining Practice and Costs of the Vipond Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, 11 pp.

[16] Park, J. F., Mining Practices, Methods, and Costs at Elkoro Mines, Jarbidge, Nev.: Bureau of Mines Inf. Circ. 6543, 1931, 12 pp.

[17] Johnson, N. O., Mining and Milling Methods and Costs, Hog Mountain Gold Mining & Milling Co., Alexander City, Ala.: Bureau of Mines Inf. Circ. 6914, 1936, 23 pp.

TABLE 36.—*Timber consumption in stopes* —Continued

III. CUT-AND-FILL STOPING

Mine	Variation of stoping method	Timber consumption per ton of ore mined		Remarks
		Board feet	Linear feet, round timber	
Matahambre, Cuba [18]	Horizontal on drift pillars	0.802		Plus 0.125 piece of prop timber.
Pecos, N. Mex. [19]	Horizontal on drift pillars and square-set stoping.		2.8 3.9	1928. 1929.
United Verde, Ariz.[20]	Horizontal on drift pillars	6.95		
Champion, Mich.[21]	Sublevel inclined	3.1		
Teziutlan, Mexico [22]	Horizontal on drift pillars	5.547		Some open stoping.
Park City Consolidated, Utah[23]	Rill stoping	5.86		
IV. SQUARE-SET STOPING				
United Verde, Ariz.[20]	Inclined square setting	12.52		Square-sets used mostly for mining level pillars.
United Verde Extension, Ariz.[24]	Vertical sections	16.82		
Tintic Standard, Utah [25]	do	19.74		Plus 4.29 board feet for maintenance.
Black Rock, Mont.[26]	Horizontal floors, some underhand vertical sections.	12.00	2.8	
Eagle, Colo.[27]	Horizontal floors	13.90		
Bawdwin, Burma [28]	Inclined	15.36		

V. TOP SLICING

Mine [3]	Variation of slicing method	Slice covering		Timber consumption per ton of ore			
		Side	Bottom	Timber		Poles, linear feet	Lagging
				Board feet	Linear feet		
							Cord
Mesabi No. 1 [29] [30]	Parallel slicing, scraping into cars.	Hardwood boards.		2.83			
Mesabi No. 3 [29] [30]	do	Wire fencing			1.16	1.24	0.0012
Mesabi No. 4 [29] [30]	do	do	Hardwood boards.	1.06		1.256	.0018
Mesabi No. 5 [29] [30]	do	do	do	1.34			.0016
Mesabi No. 6 [29] [30]	Radial slicing, scraping to chutes.	do	Wire fence on poles.		.96	1.25	.0032

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[3] Eaton, Lucian, Method and Cost of Mining Hard Specular Hematite on the Marquette Range Mich.: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[18] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

[19] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[20] Quayle, T. W., Mining Methods and Practices at the United Verde Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6440, 1931, 31 pp.

[21] Mendelsohn, Albert, Mining Methods and Costs at the Champion Mine, Painesdale, Mich.: Bureau of Mines Inf. Circ. 6515, 1931, 16 pp.

[22] Herivel, E. Ph., Mining Methods and Costs at the Teziutlan Copper Mine of the Mexican Corporation, S. A., Teziutlan, Puebla, Mexico: Bureau of Mines Inf. Circ. 6736, 1933, 16 pp.

[23] Wiles, Gloyd M., Mining Methods and Costs at the Park City Consolidated Mines Co., Park City, Utah: Bureau of Mines Inf. Circ. 6880, 1936, 13 pp.

[24] D'Arcy, R. L., Mining Practice and Methods at the United Verde Extension Mining Co., Jerome, Ariz.: Bureau of Mines Inf. Circ. 6250, 1930, 11 pp.

[25] Wade, J. W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Bureau of Mines Inf. Circ. 6360, 1930, 21 pp.

[26] McGilvra, D. B., and Healy, A. J., Methods and Cost of Mining at the Black Rock Mine, Butte & Superior Mining Co., Butte District, Mont.: Bureau of Mines Inf. Circ. 6370, 1930, 16 pp.

[27] Borcherdt, W. O., The Empire Zinc Co.'s Operation at Gilman, Colo.: Eng. and Min. Jour., vol. 132, No. 3, Aug. 10, 1931, pp. 99–105.

[28] Calhoun, A. B., Mining Methods at Bawdwin Mine: Trans. Am. Inst. Min. and Met. Eng. (issued with Min. and Met.), vol. 69, August 1923, pp. 208–247.

[29] Jackson, Chas. F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, 296 pp.

[30] Long-ton basis.

TABLE 36.—*Timber consumption in stopes*—Continued

V. TOP SLICING—continued

Mine	Variation of slicing method	Slice covering		Timber consumption per ton of ore			
		Side	Bottom	Timber		Poles, linear feet	Lagging
				Board feet	Linear feet		
							Linear foot
Cuyuna No. 8[29,30]	Radical slicing, scraping to chutes.	None	Boards		.746		1.00
Marquette No. 9[29,30]	Parallel slicing, scraping to chutes.	Wire fencing or boards.	Wire fencing on poles.		1.121		2.585
Marquette No. 11[29,30]	Both radial and parallel slicing, scraping to chutes.	do	Plank or wire fencing on poles.	1.702			3.941
Marquette No. 12[29,30]	Parallel slicing, scraping to chutes.	do	do	1.117		1.849	4.474
Marquette No. 13[29,30]	do	do	do	1.376			3.052
Menominee No. 14[29,30].	do	Wire fencing.	Wire fencing on 2-inch plank		.525		1.493
El Bordo, Mexico[31,32]	Stringers and posts, hand shoveling.		2½ by 8-inch plank.	7.1 sawed. 4.5 round			
Coronado, Ariz.[32,33]	Inclined		2- by 12-inch plank	9.00			

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[29] Jackson, Chas. F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, 296 pp.

[30] Long-ton basis.

[31] Mechin, R. J., Top-Slicing in Old Fills at El Bordo Mine, Mexico: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 139–145.

[32] Short-ton basis.

[33] Mosier, McHenry, and Sherman, Gerald, Mining Practice of Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Bureau of Mines Inf. Circ. 6107, 1929, 33 pp.

COMPARISON OF STOPING METHODS

The foregoing descriptions and discussion of the different stoping methods and practices and the accompanying tabulated data indicate the conditions under which they may be applied and serve as a basis for comparing them. Stoping costs will be presented later.

Direct comparisons of the different methods are difficult because these methods and variations thereof are applied to different conditions. Thus, for example, while it is generally true that the cost per ton by shrinkage stoping is less than by cut-and-fill stoping and that timber consumption per ton of ore mined is generally less with cut-and-fill stoping than with square-set stoping, it does not necessarily follow that shrinkage is preferable to cut-and-fill in the first instance or that cut-and-fill is preferable to square-setting in the second, because conditions might be such that shrinkage or cut-and-fill could not be used. In a given instance any supported-stope method by which support will fail before the ore can be removed is immediately out of consideration. Likewise, caving methods cannot be considered if the ore will not cave and in doing so break small enough so that it can be taken out, or if, in top slicing, the capping will not cave and follow down as the ore is removed from below.

Stoping methods may be compared broadly on the bases of applicability, flexibility, amount of preparatory development required, percentage recovery of the total ore in the deposit, dilution of the ore with waste, rate of extraction, cost of stoping, and effect upon recovery in the ore-dressing plant. Applicability of the different methods or the conditions to which each is suited have been discussed briefly in describing them.

FLEXIBILITY

Open stopes are flexible in that irregularities in the ore body usually can be followed and the ore can be mined, lean ore and waste inclusions within the ore body can be left in place as pillars and if desired, waste usually can be sorted and thrown to one side in flat stopes or piled on stulls covered with lagging in steep stopes, except those worked by underhand benching. The ore can be removed from the stopes as broken, and usually is; but often, especially in sublevel stoping, considerable broken ore can be accumulated in the stopes to form a reserve.

Shrinkage stoping lacks the flexibility of other open-stope methods. Waste cannot be sorted from the broken ore in the stopes, although some sorting may be done at the level or in grizzly chambers protected by overhead pillars. Narrow tongues or offshoots of ore cannot be followed into the walls without breaking considerable waste with the ore, and irregular walls, rolls, or sharp offsets in the vein usualy entail leaving ore behind, or considerable dilution, or both. More than half (usually about 60 percent) of the broken ore remains in the stopes until they have been mined to the top and, when finally drawn, must be taken as it comes without any control over its grade; control for grading purposes can then be secured only by drawing from several stopes of different average grade to maintain a normal mine average.

Cut-and-fill stoping is a flexible method; irregular walls can be followed and stringers or offshoots of ore can be followed and mined into the walls. Waste broken in mining these stringers can be sorted out and left in the fill. Waste inclusions within the ore body can be shot down into the fill if they are small or left in place if large. Waste broken with the ore in normal stoping operations can be sorted and left as fill in horizontal cut-and-fill stopes, though only a limited amount of sorting is possible in rill stopes. Veins of less than stoping width can be mined by stripping or resuing with minimum loss of ore and dilution of the ore with waste. Cribs or stulls can be erected on top of the fill to support local patches of loose back or wall rock, and change from cut-and-fill to square-set stoping can be made quickly after the stope has been started, should ground conditions require it. The method does not have flexibility from the standpoint of being able to force production, as breaking is interrupted periodically for removal of the broken ore and introduction of filling.

Square-set stoping is a flexible method; it can be accommodated to very irregular ore bodies, large or small, and permits following and mining offshoots of ore with minimum dilution and loss of ore. Waste can be sorted out and left in the gob. The method can be adapted where sudden changes in dip of the deposit occur and to following sinuosities or abrupt offsets along the strike or the dip. It usually is done from the bottom upward, or "overhand," but can also be employed for mining downward from the level (underhand square setting). It is used frequently as an auxiliary to other methods, as for mining pillars between filled or caved stopes, crown pillars over completed stopes that have been filled, or sections of cut-and-fill stopes where the ground is too heavy to be mined by cut-and-fill. As with cut-and-fill stoping, it does not lend itself to forcing of production.

Block caving is limited in its applicability to large deposits that cave readily and break up well as the caved mass moves downward. It is applicable only to nonselective mining, and waste inclusions within the ore body together with tongues of waste projecting into the ore body are necessarily broken and drawn with the ore. Thus, the grade of ore can be controlled only to the extent possible by regulation of the amount of capping and marginal-grade wall rock that is drawn with the ore.

Sublevel caving lacks the flexibility of supported-stope methods but is more flexible than block caving. The sublevel development serves to explore the ore body at each caving horizon before mining begins, and irregular boundaries can be followed more easily than with block caving and with less dilution from the walls. Large inclusions of waste within the ore body often can be left and caved into the gob.

Top slicing is a limited selective method in that irregular ore boundaries may be followed and the ore mined from embayments or lobes of ore extending from the main ore mass into the country rock if these lobes are large enough so that the capping over them will cave tight after a slice of ore has been removed from below it. Some sorting of waste can be done in the stopes, and waste or lean ore inclusions too large to break up readily with the caving overburden and too small to stand permanently as pillars must be broken down with the ore. Such inclusions, if left, often will cause the gob or mat to hang up for some time, and later they may let go suddenly and endanger the workmen in active slices below. If a good mat is maintained little dilution of the ore by caved capping will result. If it becomes necessary to curtail or stop production for a time the stopes can be kept in good condition for quick resumption of work by completing active slices, so that no small pillars or stubs of ore are left between them, and by boarding them up and blasting the gob down onto the floors.

Open-stope methods are adaptable only to deposits having strong ore and wall rock; hence, open-stope mines usually can be left to stand idle for considerable periods without much preparation and work resumed quickly on short notice. Shrinkage stopes and cut-and-fill stopes may require some preparation for temporary shut-downs, depending on conditions. It may be necessary to stull the backs from the top of the broken ore or of the fill. Square-set stopes, which usually are employed only in heavy ground, may require considerable work to leave them in shape for quick resumption of operations; all sets not required to be left open usually should be filled, and considerable diagonal bracing, liner sets, and blocking may be necessary. In block caving, drawing of a block should be continuous after it has once started, to avoid packing of the broken ore and thus obtain best results. Some ores tend to pack more than others, and therefore the results of a prolonged shut-down are more serious in some mines than in others. Continued drawing may also be necessary to avoid development of excessive pressure on the grizzly level and pillars.

PREPARATORY DEVELOPMENT

The subject of preparatory development has been discussed in a previous section under the caption "Stope Development" (p. 150), and tables 19 to 22, inclusive, presented data on the amount of stope development required per ton of ore developed for a number of mines representative of the various stoping methods.

With open stopes in ore bodies of the Tri-State and Southeast Missouri districts, virtually no development is required once the ore body has been cut, and the ore is stoped as developed.

Sublevel stoping requires considerable preparatory work in the form of entrance raises, starting raises or stopes, sublevel drifts, and in some instances the cutting or bulldozing chambers and installation of grizzlies. The sublevel development, however, serves thoroughly to explore and block out the ore body and to define its boundaries. By using wide shrinkage stopes instead of raises for starting each stope, production of ore need not be delayed unduly by the preparatory work. Other open-stope methods require but little preliminary work, and stoping can be started soon after a small amount of drifting has been done unless a retreat system of mining is employed.

With shrinkage stoping in narrow ore bodies, the development is usually simple and only a small amount of preparatory work is required preliminary to beginning stoping operations. Where the stopes can be supported on stulls over the drift, stope development consists of taking down drift backs and installing stull timbers and chutes; raises to the level above may be required for ventilation, but these generally can be started from the stope after it has been mined up several slices and connected through while the stope is being worked. Where the stopes are supported on drift pillars, chute raises must be driven at frequent intervals (10 to 20 feet, usually) and connected over the tops to form the stope sill. In this instance it may be desirable to funnel the tops of the chute raises before regular stoping is begun. In some mines bulldozing chambers are cut and grizzlies installed over the chute raises and raises are run up from the chamber and connected over a pillar left above the chamber, which further increases the time and work required to prepare the stope for production. On the whole, however, shrinkage stoping as usually practiced permits the starting of stopes soon after the drifts have entered the ore. As only about 35 to 40 percent of the ore broken is available until completion of a stope, more stopes are required to produce a given tonnage at the beginning of operations than otherwise would be needed. Once the mine has reached its production stride, the deficiency in productive capacity of operating stopes can be made up by drawing from completed stopes.

With cut-and-fill stoping, preparatory work is similar to that for shrinkage stoping, except that raises must be put through to the level above before stoping can progress far, if filling material has to be supplied from sources outside the stope. Partly offsetting this, however, is the smaller number of chutes and chute raises required; and they may be 25, 50, or even 100 feet apart, depending largely on the method of handling the ore in the stopes, whereas chutes for shrinkage stopes must be closely spaced and seldom are more than 20 feet apart. In flat veins it may be necessary to drive drifts in the hanging wall for introduction of fill in addition to the regular ore-extraction drifts in or along the footwall.

Development in preparing for square-set stoping is simple, but usually at least one through raise to the level above is required for each stope section to provide for introduction of fill, improve ventilation, and afford a means for lowering timbers into the stope. As soon as a raise connection is completed, however, stoping may be begun.

Block caving requires a large amount of preparatory work, and before actual extraction of caved ore begins, considerable capital must be expended for this purpose. Thus, the estimated cost of preparatory work for a block containing 120,000 tons of ore at the Morenci Branch, Phelps Dodge Corporation, was $36,420.[51] Similar costs for preparing a block of ore containing 560,000 tons at Ray were given as $81,939.[52] Comprising the preparatory work were 325 feet of supply drifts, 1,456 feet of grizzly drifts, 700 feet of haulage drifts, 1,848 feet of haulage raises, the installation of 56 grizzlies, 560 feet of excavation for pony sets and chutes, and the removal of 27,476 tons of ore from shrinkage and undercutting stopes. It is obvious that considerable time is required to complete this amount of work and that the beginning of ore production is retarded accordingly.

PERCENT RECOVERY OF ORE AND DILUTION

Recovery and dilution usually are interrelated; with some methods of stoping a high recovery involves contamination of the ore from the walls or capping, and often clean ore can be obtained only by leaving some ore in the mine.

In open-stope mines the greatest loss of ore is that tied up in pillars left for support of the back or hanging wall. The amount of ore thus left varies considerably with the strength of ore and wall rocks, thickness of the ore, its depth below surface, and the requirements for permanent support to prevent movement and subsidence of overlying strata and of the surface. Where eventual subsidence is permissible, much of the pillar ore often can be recovered on the retreat by robbing operations, the amount recoverable depending to a considerable extent on roof characteristics; sometimes almost complete extraction is possible. Where it is imperative to prevent surface subsidence and resultant damage to surface improvements or to prevent an influx of surface or underground water, 20 to 40 percent of the ore in the deposit may have to be left as pillars. Aside from ore left in pillars, high extraction is usually possible in open stopes, since the stope boundaries are exposed to view and all showings of ore can be attacked and mined.

With shrinkage stoping, high extraction can be obtained if the lode is regular in strike, dip, and thickness and the walls are firm. If the lode is very irregular, there are numerous rolls in the footwall, or there are abrupt offsets due to strike or diagonal faults or if tongues of ore make off into the walls, considerable ore may be lost. Ore in the footwall in the form of bulges or branching tongues is likely to be overlooked because covered by broken ore; however, if the walls are exceptionally firm much of this often can be recovered during final drawing of the stope. When the stopes are silled above the level, that is, on drift pillars (called boxhole pillars in Canada), complete recovery of these pillars is frequently impossible. Likewise, some loss may be incurred in recovery of floor or crown pillars. Loss in pillars may run up to 10 or 15 percent of the total ore in the deposit. As in open stoping, pillars may also be left at various horizons in the stope for roof support, but the aim usually is to leave these in low-grade sections as much as possible.

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[51] Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Bureau of Mines Inf. Circ. 6107, 1929, p. 33.

[52] Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Bureau of Mines Inf. Circ. 6167, 1929, p. 27.

Some dilution almost always occurs in shrinkage stoping, either because of overbreaking the walls of the vein during mining or later sloughing, especially during final drawing. Overbreaking is almost unavoidable when the vein is irregular in strike and dip and especially so if there are numerous small offsets.

From 85 to 95 percent of the ore usually can be recovered by shrinkage stoping with dilution of 5 or 10 to 25 percent, except where some other method would be better suited to the conditions. Sometimes dilution runs 30 percent or more, but in most such instances cut-and-fill or some other method probably should be employed.

The use of cut-and-fill stoping under conditions to which it is suited usually results in high recovery and little dilution, except in very heavy ground, as at the Morning mine, although some ore may be left in drift pillars where the stopes are silled above the level and in floor pillars. As previously stated, the fact that irregularities in the vein and tongues of ore can be mined and blind lodes prospected for in the walls, and, when found, mined as a tributary stope, makes a high recovery possible. Dilution is small, since the fill prevents sloughing of the walls, and any wall rock that comes in or horses of waste in the vein are left in the fill. Hand sorting is often employed to remove coarse gangue material with resultant raising of the grade of the ore trammed.

A high recovery with little dilution usually can be obtained with square-set stoping. This method finds its best application for mining in heavy ground where the ore is high-grade and any loss or dilution is to be avoided, regardless of cost. By square-set stoping and close filling, high-grade sections of an ore body below or adjacent to lower-grade ore often can be mined without disturbing the lower-grade sections, which changes in economic conditions or improved technology later may make it possible to mine profitably.

Recovery in block caving varies with conditions and is influenced by the amount of dilution the ore can stand, since high recovery usually involves high dilution also, and the amount of dilution permissible depends on metal prices. Thus, if the minimum grade of ore that it is profitable to handle is 1 percent copper, drawing should stop when, owing to dilution, the grade falls below this figure, regardless of whether or not all the ore has been recovered. Some dilution always results from "piping" of broken capping through the broken ore and from the margins, and the tonnage drawn from a block of ore usually is greater than the estimated tonnage in the block, whereas the average grade of ore usually is less than that estimated. If the margins of the deposit are very irregular in outline, either the extraction or the grade of ore recovered will suffer, as it is impossible to make the cave follow an irregular boundary closely. At Ray, Ariz., the dilution in drawing 40,000,000 tons of ore was estimated as 10 percent. At Inspiration, 110 percent of the estimated tonnage and more than 85 percent of the metallic contents are recovered. At Miami, operations in 13 completed stopes or blocks resulted in the recovery of 2.4 percent more copper than had been estimated previously, but 15.15 percent more tonnage was drawn, the average grade being 88.93 percent of the calculated grade. Careful regulation of drawing is essential to obtain minimum dilution and maximum recovery.

With sublevel caving the line of break may be controlled more closely around the boundaries than with block caving, as the ore is partly excavated by slicing, which can be confined strictly to the ore, and the blocks caved are relatively small both in vertical height and in horizontal area. However, as the backs are caved under a gob consisting of a mixture of broken capping, slice timbers, and flooring, which cannot always be prevented from breaking through and cutting off undrawn ore, some ore is bound to be lost; recovery usually is not as complete as with top slicing. Under the conditions on the Gogebic iron range, Michigan, where much of the capping carries 40 percent or more iron, a small amount of dilution does not seriously affect the grade of ore mined and the dilution from this source may offset tonnage of ore lost; the net result is therefore likely to be approximately 100 percent tonnage recovery, with a slight lowering of the total contained iron below that of the ore body proper.

Top slicing permits high recovery of ore with little dilution (provided a suitable mat is maintained) under conditions where any method of mining from the bottom upward would be difficult, hazardous, and costly. Small stubs of ore between the ends of slices or between the end of a slice and the cave are sometimes lost, and 2 or 3 feet of ore may have to be left around the margins of a deposit where a flat footwall comes up to the overlying mat.

Accurate figures on recovery and dilution are difficult to obtain unless waste and ore faces are measured and sampled after each round and the waste sorted out in the stopes is measured accurately. The finding during stoping of additional ore in the form of bulges in the lode or offshoots therefrom, inclusions of waste within the ore body, the sorting of waste from the broken ore, and the admixture with the ore, from sloughing, of waste and of capping or walls containing some ore minerals may combine to obscure the true situation with regard to recovery and dilution. Sometimes low-grade ore not included in the original estimates is found in mining under conditions such that it is profitable to extract it, and, on the other hand, low metal prices may necessitate leaving considerable ore that would be mined under higher prices.

It thus appears that "dilution" and "recovery" as applied to stoping results are not specific or rigid terms, and the data on recovery and dilution in table 37 should be interpreted with this in mind. In circulars dealing with cut-and-fill and square-set stoping, the authors have usually stated or implied that complete recovery is obtained and no figures are given; this accounts for the small amount of data on these methods of stoping in the table.

TABLE 37.—*Recovery and dilution data*

Mine	Stoping method and variation	Estimated recovery of total ore, percent	Estimated dilution, percent	Remarks
Tri-State No. 1[1]	Open stoping, casual pillars.	[2] 85	None [3]	Final recovery expected to be 90 percent and most of remaining 10 percent lean ore.
Tri-State No. 2,[4]	do	[2] 75	do.[3]	Final recovery expected to be 80 percent.
Barr, Kans.[5]	do	[2] 82–85	do.[3]	Most of pillars expected to be recoverable in final clean-up.
No. 8, Southeast Missouri.[6]	do	[2] 88	do.[3]	
Crestline, Kans.[7]	do	[2] 78–100	do.[3]	
Hartley-Grantham, Kans.[8]	do	[2] 60	do.[3]	Final clean-up expected to result in nearly 100 percent extraction.
Marquette, No. 1, Mich.[9].	Open stoping, regular pillars.	[2] 72		Part of mine under valuable surface improvements; 90 percent recovery in narrow ore.
Burra-Burra, Tenn.[10]	Sublevel stoping	85	5	
Marquette No. 2, Mich.[11]	Sublevel stoping (⅓) and topslicing (⅔).	90	10	
Menominee No. 1, Mich.[12]	Sublevel stoping	[2] 75		Some of pillars will ultimately be extracted.
Spring Hill, Mont.[13]	Modified sublevel stoping.	[2] 90	Practically none.	
Edwards, N. Y.[14]	Open stoping, with casual pillars. Shrinkage stoping	[2] 85–90	30 Less than 30	Dilution rock carries considerable zinc. Tonnage hoisted greater that estimated as ore.
Presidio, Tex.[15]	Open stopes with casual pillars.	[2] 75	Practically none.	6 percent of broken rock sorted out as waste.
Granada, Que.[16]	Open stopes, overhand	[2] 98	35–50	Wall rock contains some gold.
Mt. Isa, Queensland.[18]	Sublevel stoping	[2] 60		Pillars to be extracted later.
Engels, Calif.[17]	Shrinkage stoping on drift pillars. Pillars between large stopes.	[2] 96. 4		

[1] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 1 Mine, Tri-State Zinc and Lead District, Picher, Okla.: Bureau of Mines Inf. Circ. 6113, 1929, 11 pp.

[2] Balance in pillars.

[3] Most of overbreak contains some ore mineral, and final pillars are usually in low-grade ore.

[4] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.: Bureau of Mines Inf. Circ. 6121, 1929, 11 pp.

[5] Keener, O. W., Method and Cost of Mining at Barr Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6159, 1929, 9 pp.

[6] Poston, R. H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Bureau of Mines Inf. Circ. 6160, 1929, 22 pp.

[7] Netzeband, W. F., Method and Cost of Mining Zinc and Lead at No. 3 Mine, Tri-State District, Crestline, Kans.: Bureau of Mines Inf. Circ. 6174, 1929, 10 pp.

[8] Keener, O. W., Methods and Costs of Mining at the Hartley-Grantham Mine, Tri-State Zinc and Lead District: Bureau of Mines Inf. Circ. 6286, 1930, 8 pp.

[9] Eaton, Lucien, Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan: Bureau of Mines Inf. Circ. 6138, 1929, 14 pp.

[10] McNaughton, C. H., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn.: Bureau of Mines Inf. Circ. 6149, 1929, 17 pp.

[11] Eaton, Lucien, Mining Soft Hematite at Mine No. 2 of the Marquette Range, Mich.: Bureau of Mines Inf. Circ. 6179, 1929, 15 pp.

[12] Eaton, Lucien, Mining Soft Hematite by Open Stopes at Mine No. 1, Menominee Range, Mich.: Bureau of Mines Inf. Circ. 6180, 1929, 10 pp.

[13] Pierce, A. L., Mining Methods and Costs at the Spring Hill Mine, Montana Mines Corporation, Helena, Mont.: Bureau of Mines Inf. Circ. 6402, 1931, 11 pp.

[14] Knaebel, J. B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, N. Y.: Bureau of Mines Inf. Circ. 6586, 1932, 25 pp.

[15] Howbert, Van Dine, and Bosustow, Richard, Mining Methods and Costs at Presidio Mine of the American Metal Co. of Texas: Trans. Am. Inst. Min. and Met. Eng., 1931, pp. 38–50.

[16] Loofbourow, R. F., Mining Methods and Costs at Granada Gold Mines, Ltd., Rouyn, Quebec: Bureau of Mines Inf. Circ. 6709, 1933, 15 pp.

[17] Kruttschnitt, J., and Mann, V. I., Mining Methods and Costs at Mount Isa Mines, Ltd., Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

[18] Nelson, W. I., Mining Methods and Costs at the Engels Mine, Plumas County, Calif.: Bureau of Mines Inf. Circ. 6260, 1930, 22 pp.

TABLE 37.—*Recovery and dilution data*—Continued

Mine	Stoping method and variation	Estimated recovery of total ore, percent	Estimated dilution, percent	Remarks
Nevada-Massachusetts, Nev.[19]	Shrinkage stoping on drift pillars.	[2] 95	20	
Consolidated Cortez, Nev.[20]	do	[2] 95		
Eighty-five, N. Mex.[21]	Shrinkage stoping on drift timbers.		15–18 or more	
Verde Central, Ariz.[22]	Shrinkage stoping on drift timbers; retreat mining.		10	
Kirkland Lake, Ont.[23]	Shrinkage stoping on drift timbers.		30	
Vipond, Ont.[24]	Shrinkage stoping; partly on pillars and partly on timbers.	98	10	
Mt. Hope, N. J.[25]	Shrinkage stoping on slope pillars.	[1] 90	10	
Matahambre, Cuba[26]	Horizontal cut-and-fill; some square setting.	90		On account of cost of timber some low-grade ore deliberately left in square set stopes.
Morning, Idaho[27]	Stringer-set-and-fill		20	
Pecos, N. Mex.[28]	Mostly horizontal cut-and-fill stoping.	Nearly 100		
McIntyre, Ont.[29]	Shrinkage stoping and horizontal cut-and-fill on drift timbers.	110–115	10–15	
Tintic Standard, Utah.[30]	Square-set stoping, vertical sections.	99+		
Ray, Ariz.[31]	Block caving	110	?	
Inspiration, Ariz.[32]	do	110+	85+	Recovery of estimated metal content.
Miami, Ariz.[33]	do	115.15	102.4	Grade 88.93 percent of estimated grade. Figures cover 13 completed stopes and tonnage drawn was 12,710,378.

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[2] Balance in pillars.

[19] Heizer, O. F., Method and Cost of Mining Tungsten Ore at Nevada-Massachusetts Mines at Mill City, Nev.: Bureau of Mines Inf. Circ. 6284, 1930, 13 pp.

[20] Hezzelwood, G. W., Mining Methods and Costs at the Consolidated Cortez Silver Mine, Cortez, Nev.: Bureau of Mines Inf. Circ. 6327, 1930, 15 pp.

[21] Youtz, R. B., Mining Methods at the Eighty-Five Mine, Valedon, N. Mex.: Bureau of Mines Inf. Circ. 6413, 1931, 26 pp.

[22] Dickson, R. H., Methods and Costs of Mining Copper Ore at the Verde Central Mines (Inc.), Jerome, Ariz.: Bureau of Mines Inf. Circ. 6464, 1931, 13 pp.

[23] Dumbrille, J. C., Mining Methods of Kirkland Lake Gold-Mining Co. (Ltd.), at Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6490, 1931, 12 pp.

[24] Dye, R. E., Mining Practice and Costs of the Vipond Mine, Timmins, Ontario: Bureau of Mines Inf. Circ. 6525, 1931, 11 pp.

[25] Sweet, J. R., Mining Methods and Costs at the Mount Hope Mine of the Warren Foundry & Pipe Corporation, Mount Hope, N. J.: Bureau of Mines Inf. Circ. 6601, 1932, 31 pp.

[26] Richert, G. I., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Bureau of Mines Inf. Circ. 6145, 1929, 18 pp.

[27] Wethered, C. E., and Coady, L. J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6238, 1930, 13 pp.

[28] Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.

[29] Skavlem, H. G., Mining Methods and Costs at the McIntyre Porcupine Mines, Ltd., Schumacher, Ontario: Bureau of Mines Inf. Circ. 6741, 1933, 19 pp.

[30] Wade, J. W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Bureau of Mines Inf. Circ. 6360, 1930, 21 pp.

[31] Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Bureau of Mines Inf. Circ. 6167, 1929, 27 pp.

[32] Stoddard, A. C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Bureau of Mines Inf. Circ. 6169, 1929, 23 pp.

[33] Maclennan, F. W., Miami Copper Co. Method of Mining Low-Grade Ore Body: Am. Inst. Min. and Met. Eng. Tech, Pub. 314, 1930, 44 pp.; *also* Trans. Am. Inst. Min. and Met. Eng., vol. 91 (Yearbook), 1930, p. 64.

RATE OF EXTRACTION

Some stoping methods are inherently slow, whereas others permit rapid extraction of ore from the stope and, as previously stated, permit flexibility in rate of output. Thus, in cut-and-fill stoping the production rate cannot be forced very much as ore-breaking must be stopped in sections being filled, although production from an entire stope may be constant by breaking on one side of the ore pass while filling on the other side. In square-set stoping production, rate is slow (1) because the ore is broken in small blocks that usually can be drilled off in a short time, whereas timbering and filling consume a large part of the working shift and (2) because production cannot be forced in heavy ground without inviting danger of collapsing the stope. In open stopes where there are large areas of stope faces, production often can be forced by simply putting more machines to work. In sublevel stoping production can be speeded by simultaneous operations on all the sublevels and often by increasing the number of machines on each bench.

Forced caving, as employed at Alaska-Juneau, Beatson, and Climax, and block caving—all nonselective methods—permit a high production rate once ore production begins, although considerable time is consumed in preparatory work, especially for block caving. Best results are obtained with block caving if the ore is drawn at a uniform rate; therefore, any forcing of this rate is likely to cause the piping through of waste and incomplete breaking up of the ore.

The rate of output possible with any stoping method depends to a large extent upon the size of the ore body, especially the width, and the width of the stopes; a high rate of output is impossible in narrow stopes.

EFFECT ON RECOVERY IN ORE-DRESSING PLANT

The stoping method may affect the percentage of ore minerals recovered in the ore-dressing plant, especially where certain sulfide ores are concentrated by flotation. Thus, a stoping method that permits immediate removal of the ore after it is broken so as to preclude its partial oxidation before it goes to the mill may be desirable or necessary. Under these conditions, shrinkage stoping and block caving, in which large masses of broken ore are necessarily left in the mine for a considerable length of time, may be unsuitable.

COST OF STOPING

Stoping cost is one of the largest of the items comprising the total cost of producing ore. The cost per ton of ore mined is important but may be overemphasized if the more important cost, that per unit of metal recovered, is thereby overlooked. A given stoping method may give a low cost per ton of ore but, by reason of high dilution involving the handling, hauling, hoisting, and treating of a large amount of barren rock, may result in a high cost per unit of metal. Low cost per ton may involve loss of ore or excessive dilution with waste and a low-cost-per-ton method may not, therefore, be the most economical one in a given instance.

Table 38, giving stoping costs per ton of ore mined, should be considered with the foregoing important fact in mind. The tabulated costs do not include any development, haulage, hoisting, or pumping unless so indicated but are direct-stoping costs only. Wide differences, as between mines, in the distributed costs shown in the tables are partly explained by differences in accounting methods, but the total stoping costs in the last column are believed to be comparable except where otherwise noted.

TABLE 38.—*Stoping costs*

I. OPEN STOPES

Mine	Year	Variation of open-stope method	Direct stoping costs per ton of ore mined							
			Labor	Supervision	Air compression, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Average of 7 mines, Tri-State district.	1927–30	Irregular pillars; flat beds	$0.273	$0.020	$0.102	$0.005	$0.104			$0.504
No. 8, Southeast Missouri.	1928	do	.368		.033	.010	.072			.483
Mascot, Tenn	1929	Mill-hole system irregular pillars	.130		.042		.045		.005	.222
Marquette range, No. 1 hard-ore mine, Mich.[1]	1928	Regular room and pillar	.514	.028	.104	.006	.105	$0.002	.011	.770
Mineville, N. Y.[1]	1927	Irregular pillars	.164	.014	.113		.122		.003	.416
Osceola, Mich	1931	Wide stopes, irregular pillars								[2].456
Ahmeek, Mich	1931	Long narrow stopes; regular narrow pillars								[2].589
Vanadium, Colo	1931	Room-and-pillar; pillars extracted on retreat.	.740	.050	.090	.130	.200	.040	.100	1.350
Granada, Que	July 1932	Overhand mining; no pillars except in lean ore and waste.	1.556	.087	0.476		.606	.040		[3] 2.765
Mary, Tenn	1928	Underhand mining; regular stope and pillar.	.509	.022	.177		.076			.784
Burra-Burra, Tenn	1928	Sublevel stoping, pillar mining, and some shrinkage.	.196		.051		.083		.002	.332
Do	1932	Sublevel stoping	.100		.081		.078			.259
No. 1, Menominee range, Mich.[1]	1928	Sublevel stoping, vertical stope face	.246	.031	.080		.090	.009	.011	.467
No. 2, Marquette range, Mich.[1]	1928	Sublevel stoping, outward sloping stope face.	.180	.025	.030	.047	.050	.010	.075	.417
Spring Hill, Mont	Aug. 1, 1929, to Apr. 30, 1930.	Sublevel stoping in wide ore; shrinkage stoping in narrow ore.	.738		.124		.178		.032	1.072
Sherritt-Gordon, Man.[4]	1931–32	Underhand benching similar to sublevel stoping on dips above 45°, but benches 150 feet high measured on dip.	.298		.150		.210		.089	.747
Do	1932	Overhand, using scrapers on dips under 45°	.302		.164 [5].054		.146		.074	.740
Hanover-Bessemer, N. Mex.	1930	Underhand benching to inclined stope raises.	.167	.010	.075		.086	.003	.002	.343
Hartley, Kans	1932 (6 mos.)	Open stopes, irregular pillars	.302		.078	.031	.150		.011	.572
Presidio, Tex.[6]	1929	do	1.442	.146	.430		.177	.025	.056	2.276
Teziutlan, Mexico	July 1930 to June 1931	Open stope and pillar; cut-and-fill	.877	.041	.252		.216	.107	.078	1.571
El Potosi, Mexico	Aug. 1933	Underhand benching	.180		.009	.058	.052	.010	.028	.337
Pilgrim, Ariz.[7]	Last quarter 1936	Open stoping, stull support	.722		.147				[8].283	1.152

Mt. Isa, Queensland	July 1935 to June 30, 1936	Sublevel stoping	.639	.034	.181	.030	.208	.043	.076	1.211
Aldermac, Quebec [4]	1938	do								.780
Beattie, Quebec [9]	1937	Spiral stoping								.610
Canadian Malartic, Quebec.[10]	1938	Sublevel stoping	.336	.032	.132		.146	.003	.038	.687

II. SHRINKAGE STOPING

Harmony, Idaho	January–June 1929	On drift pillars with grizzlies over chutes	$0.650	$0.300	$0.270	$0.160	$0.270	$0.180	$0.220	$2.050
Nevada-Massachusetts, Nev.	1928	On drift pillars	1.662	.064	.100	.209	.355	.146	.207	2.743
Hillside, Ill	1929	On drift pillars in wide ore; on stulls in narrow ore.	.700	.020	.115	.087	.219	.082	.053	1.276
Daisy, Ill	1929	On stulls	.710	.065	.190	.080	.160	.120	.070	1.395
Cortez, Nev	1929	On drift pillars	1.082	.086	.250	.070	.304	.054	.126	1.972
Eighty-Five, N. Mex.:										
8-stope	1925	On stulls	.590		.390		.230	.140	.020	1.370
6-stope	1925	do	.750		.460		.270	.200	.020	1.700
26-stope	1928–29	On pillars	.720		.640		.310	.070	.010	1.750
3-stope	1927	On stulls	.490		.390		.240	.100	.010	1.230
Verde Central, Ariz	June 1929, to August 1930	do	.734	.073	.485		.234	.135		1.661
Teck-Hughes, Ontario	1928	On drift pillars	.761	.096	.102	.207	.316	.042	.033	1.557
Kirkland Lake, Ontario.	1930	On drift timbers	.770	.070	.230		.460	.030	.060	[11].620
Do	1930	do	.671	.061	.200		.400	.027	.053	[12] 1.412
Elkoro, Nev	1930	On stulls	.919		.048	.114	.107	.164	.258	[13] 1.610
Vipond, Ontario	August 1929 to July 1930, inclusive.	Partly on stulls, partly on drift pillars	.626	.084	.250		.180	.046	.089	1.275
Sylvanite, Ontario	3 months, 1930	On stulls	1.215	.105	.468		.410	.093	.027	2.318
Porcupine United, Ontario.	5 months, 1930; 17 months, 1929–30.	do	1.437	.347	.714		.305	.120		2.923
Black Butte, Oreg	7 months, 1929	On drift pillars	.950	.070		.070	.150		.070	1.310
Engels, Calif	1928	do	.378	.033	.135		.157	.017	.040	.760
Marquette No. 4, Mich	1929	Semishrinkage between stope pillars	.470	.041	.150		.100	.012	.124	[14].897
Mt. Hope, N. J	1930	On stope pillars	.223	.036	.200		.128	.012	.033	.632
Colorada, Mexico	6 months, 1931	On drift pillars	.3186		.100		.1210	.0056	.0038	.549

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[1] Costs per long ton of 2,240 pounds.

[2] Breaking cost.

[3] One-third of broken rock sorted and rejected as waste in mine. Total direct stoping cost per ton broken was $1.843.

[4] Brown, Eldon L., Mining Methods and Costs at the Sherritt Gordon Mine: Canadian Inst. Min. and Met. Bull. 257, September 1933, pp. 468–494.

[5] Scraping.

[6] Howbert, Van Dyne, and Bosustow, Richard, Mining Methods and Costs at Presidio Mine of the American Metal Co. of Texas: Trans. Am. Inst. Min. and Met. Eng., 1931, pp. 38–50.

[7] Hastings, Earle F., Mining and Milling Methods at the Pilgrim Mine, Chloride, Ariz.: Bureau of Mines Inf. Circ. 6945, 1937, 18 pp.

[8] All supplies.

[9] Tuttle, Jay, Spiral Stoping at Beattie: Canadian Inst. Min. and Met., Bull. 322, February 1939, pp. 95–122.

[10] Neelands, E. V., and Millenbach, J. P., Mining Methods at Canadian Malartic: Canadian Inst. Min. and Met. Bull. 321, January 1939, pp. 35–49.

[11] Cost per ton of ore hoisted.

[12] Cost per ton of ore broken.

[13] Stope ore only.

[14] Per long ton and includes stope development ($0.298).

TABLE 38.—*Stoping costs*—Continued

III. FORCED CAVING

Mine	Year	Variation of open-stope method	Direct stoping costs per ton of ore mined							
			Labor	Supervision	Air compression, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Alaska-Juneau, Alaska	1928	Powder drift blasting	$0.0617		[15] $0.0019		$0.0499		$0.0084	$0.1219
Beatson, Alaska	1922–1926, incl	Stopes 70 ft. wide between 30-ft. pillars. Blasting long holes around raises.	[16] .1447		.0647		.0778	$0.0020	.0768	[16] .3660 [17] .2370
Britannia, East Bluff, British Columbia.	1934?	Undercut and blast down with long drill holes.	.055	$0.013	.010	$0.010	.098	.016	.043	.235

IV. CUT-AND-FILL STOPING

Mine	Year	Variation of open-stope method	Labor	Supervision	Air compression, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Questa, N. Mex	1929–1930	Horizontal; largely resuing								[18] $5.510
Cold Springs, Colo	1931	Resuing	[18] 3.238	[18] 0.672	[18] 0.622	[18] 0.494	[18] 0.323	[18] 0.432	[18] 0.170	[18] 5.951
Do	do	do	[19] .497	[19] .103	[19] .095	[19] .050	[19] .050	[19] .066	[19] .026	[18] .914
Lucky Tiger, Mexico [20]	Nov. 1924	Mostly horizontal resuing; rill stoping in wide ore.	2.846	.900	1.228		.510	.708	.121	[21] 6.313
Eighty-Five, N. Mex.:										
10-stope	1928–30	Inclined	1.260		.600		.280	.200	.030	2.370
26-stope	1929–30	do	1.620		.380		.170	.270	.020	2.460
Matahambre, Cuba	1928	Horizontal [22]	.693	.095	.085	.008	.095	.192	.099	1.267
Teziutlan, Mexico	July 1930 to June 1931	do [23]	.877	.041	.252		.216	.107	.078	1.571
Pecos, N. Mex	1929	do	[24] 1.520		[24] .090		[24] .140	[24] .260		[24] 2.010
Calumet and Arizona, Ariz.:										
68-stope	May 1926 to Jan. 1, 1930	Inclined between double square-set leads	.620		.215		.135	.155		1.125
90-stope	1928 and 1929	Inclined between single square-set leads	.700		.200		.100	.130		1.130
Pilares, Mexico	1929	Horizontal and inclined [25]	.630	.181	.175		.134	.243	.007	1.370
United Eastern, Ariz [26]	Jan. 1917 to May 1924	Horizontal	1.750		.136		.414	.445	.178	2.923
Champion, Mich. [27]	1927	do	1.340						.423	[28] 1.763
	Aug. 1 to Dec. 31, 1930	Sublevel, inclined, 40% sorted as waste	.794	.056	.128		.103	.061	.020	1.162
Hecla, Idaho	1928	Stringer set and fill	1.030	.080	.070		.112	.357	.101	1.750
Morning, Idaho	1928	do	1.550	.142	.091	.098	.114	.322	.054	2.371
Silver King Coalition, Utah.	1929	do	2.454	.130	.472		.173	.249	.024	3.502
Colorada, Mexico	6 mos. 1931	Horizontal between stope pillars	.4766		.0996		.1025	.1138	.0059	.7984
Magma, Ariz	1928	Inclined between square-set lines	1.490	.267	.414		.150	.470	.058	2.849
McIntyre, Ontario	1931	Horizontal cut-and-fill; some shrinkage stoping.	.891	.038	.031	.054	.147	.240	.088	1.489

Park City Cons., Utah	1935	Inclined cut-and-fill on drift sets and open stulled stopes.	.889	.044	.053	----------	.075	.150	.014	1.225
Hollinger, Ontario [29]	1934	Horizontal cut-and-fill; some shrinkage stoping in narrow ore.	.8866		.1556		.1345	[30].1801	.0853	1.4421
Ground Hog, N. Mex.	3 mos. 1930	Inclined; square set and fill in wide ore	1.197	.065	.053	.034	.058	.120	.235	1.762
Golden Anchor, Idaho	1937	Horizontal, resuing	[31] 2.52	[32].31	.310		.200	.070	.370	3.78

V. SQUARE-SET STOPING

Argonaut, Calif.	September 1929	Horizontal floors	$1.746	$0.052	$0.013	$0.088	$0.148	$0.335	$0.057	$2.439
United Verde Extension, Ariz.	1928	Vertical slices 2 or 3 sets wide, 10 to 20 sets long.	----------	----------	.244		(All other, 1.715)			1.959
Bunker Hill & Sullivan, Idaho.	1928	Inverted V-shaped back	1.377	.068	.194		.154	.315	.164	2.272
Page, Idaho	1928	Horizontal	1.430	.150	.110	.050	.110	.260	.040	2.150
Pecos, N. Mex.	1929	do	2.160	----------	([33])		.150	.400	.120	2.830
Park-Utah, Utah	1928	Inclined sections	1.437	.060	.101	.091	.112	.423	.202	2.426
Tintic Standard, Utah	November 1929	Vertical sections, 2 sets wide	2.400	.254	.521		.099	.691	.059	4.024 [34] .949 4.973
Bawdwin, Burma [35]	----------	Rill sections, 50 ft. long by 3 or 4 sets wide	.792	.198	[36].027		.234	.317	.225	1.793 [34] .430 2.223
Gold Hill, Idaho	September 1936	Horizontal	[37] .990	[37].170	[37].340	[37].070	[37].090	[37].280	[37].380	[37] 2.320
Eureka Standard, Utah	February 1933	do	[38] 1.520	----------	[39].360	----------	----------	[40].730	.620	3.230
Victoria, British Columbia.	1934?	Vertical sections 6 sets long	.462	.030	.054	.071	.126	[41].388	----------	1.131
Dayton Consolidated	1936	Horizontal	1.125	.103	1.186	----------	.127	.353	.045	[42] 1.939

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[15] Power.

[16] Includes benching in stope and pillar raises, and also bulldozing.

[17] Stope preparation.

[18] Per ton of sorted ore; includes cost of sorting.

[19] Per ton of ore and rock broken.

[20] Mishler, R. T., and Budrow, L. R., Methods of Mining and Ore Estimation at Lucky Tiger Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 469–484.

[21] Per ton of ore hoisted.

[22] 70 percent cut-and-fill, 30 percent square-setting.

[23] Mostly cut-and-fill, some open-stoping.

[24] Costs cover cut-and-fill stoping only; about 20 percent of stope rock sorted out as waste in stopes.

[25] 59.4 percent horizontal cut-and-fill, 17.6 percent inclined cut-and-fill, 17.7 percent square-set stoping, 3.7 percent top-slicing, 1.7 percent shrinkage stoping.

[26] Moore, Roy W., Mining Methods and Records at the United Eastern Mine: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, pp. 56–92.

[27] Schacht, W. H., Mining Methods of the Copper Range Co.: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 346–369.

[28] Includes tramming, timbering, track, compressor operation, and drill expense.

[29] Young, Arthur W., Mining [at Hollinger]: Canadian Min. Jour., September 1935, p. 396.

[30] Includes filling material.

[31] Includes liability insurance, $0.12.

[32] Includes superintendence, assaying, engineering, and social security.

[33] Not segregated.

[34] Maintenance.

[35] Calhoun, A. B., Mining Methods at Bawdwin Mine: Trans. Am. Inst. Min. and Met. Eng. (issued with Min. and Met.), vol. 69, August 1923, 40 pp.

[36] Air and steel only.

[37] Includes development.

[38] Includes explosives.

[39] Air compression; includes air for development.

[40] Includes timber and supplies for development; maintenance and filling ($0.22).

[41] Timber $0.113 plus filling $0.275.

[42] Per ton of ore from stopes only.

TABLE 38.—*Stoping costs*—Continued

VI. CAVED-STOPE METHODS

Mine	Year	Variation of open-stope method	Direct stoping costs per ton of ore mined							
			Labor	Supervision	Air compression, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Ohio Copper, Utah	October 1911	Blocks 30 ft. high undercut and caved to finger raises.								$0.2197
Do	April 1913	do								[43].2220
Ray, Ariz.:										
Preparatory excavation.	1928	Block caving	$0.0581		$0.0010	$0.0014	$0.0070	$0.0339	$0.0022	.1036
Stoping			.0711		.0016	.0050	.0247	.0082	.0059	.1165
Drawing ore			.0940		.0001	.0001	.0190	.0129	.0006	.1267
Total			.2232		.0027	.0065	.0507	.0550	.0087	.3468
Inspiration, Ariz	1928	do	.153		.005		.0170	.0310	.0020	.2080
Miami, Ariz	Oct. 1, 1925, to Sept. 30, 1929.	do								[44].0299 [45].1363
Humbolt (Morenci), Ariz.:										
Stope preparation	1928	do	.029				.009	.021		.059
Breaking and drawing.			.085				.034	.010		.129
Undistributed				.040	.045					.085
Total			.114	.040	.045		.043	.031		.273
Braden, Chile	1928	Block caving. Undercut by widening over series of parallel timbered drifts.	.1271		.0128		.0216	.0448	.0110	.2173 [46].0620 .2793
Eureka-Asteroid, Mich.[47]	1929	Sublevel caving. Stoping	.246	.014	.067		.069		.002	.398
		Timbering [48]	.193	.011	.001			.068	.006	.279
Total			.439	.025	.068		.069	.068	.008	.677

Montreal, Wis.[47]	1929	Sublevel caving. Stoping	.224	.035	.020	------	.061	------	.020	.360
		Timbering [48]	.149	------	------	------	.007	.060	.008	.224
Total			.373	.035	.020	------	.068	.060	.028	.584
Mesabi No. 1, Minn.[47]	1928	Parallel top slicing, scraping into cars	.4257	.0386	.0259	.0230	.0606	.1411	.0140	.7289
Marquette, Mich., No. 9.[47]	1929	Parallel top slicing, scraping to chutes	.421	.018	.095	.008	.055	.079	.063	.739
Marquette, Mich., No. 10.[47]	1928	Radial top slicing, scraping to chutes	.370	.025	.045	.047	.075	.070	.075	.707
Menominee, Mich., No. 15.[47]	1928	do	------	------	------	------	------	------	------	.611
Oceanic, Calif.[49]	1935	Parallel slicing, hand shoveling to chutes [50]	[51] 1.547		0.081		.079	.141	.077	[51] 1.925
Judge, Utah [52]	6 months, 1937	Two-thirds top slicing; hand shoveling into cars. One-third stringer-set-and-fill. Sorting in both top slicing and stringer-set stopes.	2.589		.228	.138	.236	.990	.018	4.199
Calumet and Arizona, Ariz.[53]	1916	Mitchell top slice	.540		.100	------	.040	.210	.040	.930
Bordo, Mexico [54]	1925	Top slicing, hand shoveling, stringers and posts.	.450		(55)	.010	.070	.250	------	.780
Morenci, Ariz	1917	Inclined top slicing	------	------	------	------	------	------	------	.750
Colorada, Mexico	1931	Top slicing pillars between filled stopes	.5335	------	.0999	------	.1314	.2634	.0072	1.0354

VII. MINES EMPLOYING SEVERAL METHODS

Colorada, Mexico	6 months, 1929	(56)	$0.3748	$0.1316	$0.1808	$0.0071	$0.2146	$0.1979	$0.0966	$1.2034
Campbell, Ariz	1928, 1929	(57)	.765	------	0.220		.120	.175	------	1.280
Junction and Campbell, Ariz.	1928, 1929	(57)	.865	------	.160		.095	.215	------	1.335
Pilares, Mexico	1929	(58)	.630	.181	.175		.134	.243	.007	1.370

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[43] Includes development and equipment.

[44] Preparatory work above grizzly level.

[45] Drawing and maintenance.

[46] General expenses.

[47] Costs per long ton.

[48] Includes timber for development.

[49] Frolli, A. W., Mining and Reduction Methods and Costs at the Oceanic Quicksilver Mine, Cambria, San Luis Obispo County, Calif.: Bureau of Mines Inf. Circ. 6950, 1937, 13 pp.

[50] 80 percent top slicing; 20 percent square-set stoping.

[51] Includes compensation insurance, $0.121.

[52] Krueger, Geo. S., and Hewitt, E. A., Mining Methods and Costs at the Judge Mine, Park City, Utah: Bureau of Mines Inf. Circ. 7003, 1938, 10 pp.

[53] Wilson, Philip D., Comparison of Stoping Methods at Calumet and Arizona Mine: Min. and Sci. Press, Aug. 26, 1916, pp. 315-317.

[54] Mechin, R. J., Top-Slicing in Old Fills at El Bordo Mine, Mexico: Trans. Am. Inst. Min. and Met. Engr., vol 72, pp. 139-145.

[55] Included in other costs.

[56] Rill cut-and-fill, 104,057 tons; horizontal cut-and-fill, 37,949 tons; square-set, 32,220 tons; shrinkage, 36,354 tons; drifting, 51,696 tons.

[57] Inclined cut-and-fill and semishrinkage.

[58] Horizontal cut-and-fill, inclined cut-and-fill, square-set stoping, shrinkage stoping, top slicing.

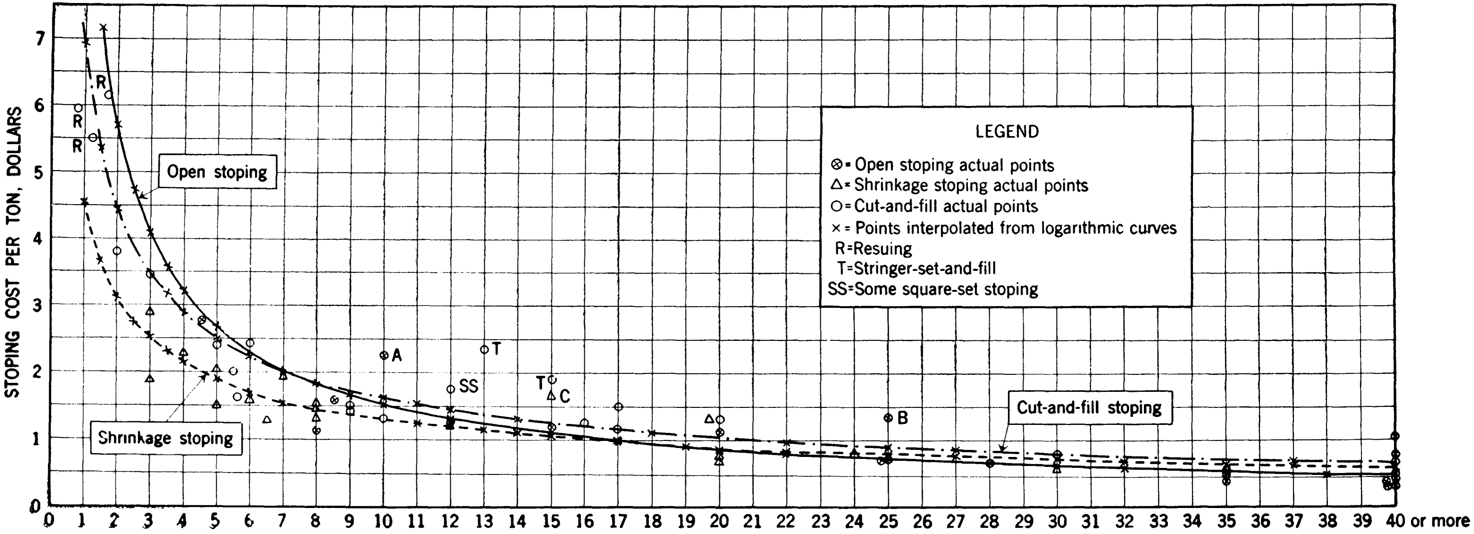

Figure 119.–Curves indicating effect of stoping widths on direct stoping cost

In figure 119, direct stoping costs are plotted against approximate vein or stoping widths, in feet, for a number of mines employing, respectively, open stoping, shrinkage stoping, and cut-and-fill stoping, and thus indicate the effect of width upon direct stoping cost per ton of ore. High costs in narrow stopes are due principally to reduction in tons broken per foot of hole drilled and hence per man-shift.

Analysis of the curves emphasizes other factors affecting the cost. Thus, at *A* a high-cost open-stope operation is shown, caused by great irregularity of ore outline, numerous displacements of the ore body by faulting, and mining of much thin ore around the margins. At *B*, a higher than normal cost may be attributed to the fact that although the stopes are about 25 feet wide, much of the ore is very thin. At *SS*, *T*, and *T* are shown higher than normal cut-and-fill stoping costs. *T* and *T* represent mines where the stringer-set-and-fill method, using large quantities of timber, is employed under very heavy hanging-wall conditions that require cautious mining. At *SS* the higher cost is due to the use of considerable square-set stoping in conjunction with cut-and-fill stoping.

The effect of resuing, involving the breaking of considerable wall rock and resulting in low tonnage per man-shift, and added costs for sorting in the stopes, is indicated by the costs *R–R–R*.

The points used for determining the shrinkage-stoping curve are seen to be quite erratic in their distribution, few of them actually falling on the approximate mean curve. This may be interpreted to show that with shrinkage stoping, in particular, other factors such as hardness of ore, tonnage mined per month or year (affecting supervisory and other distributed costs), and the physical peculiarities of individual ore bodies affect the costs to an extent that clouds the effects of width. The high cost shown at *C* is due to irregularity of ore bodies, the use of some cut-and-fill stoping, and the necessity for considerable secondary blasting.

In general, the curves indicate that stoping costs vary considerably with the width for widths of less than about 8 to 10 feet, whereas for greater widths they are influenced only slightly by the stoping width.

Costs of square-set stoping (not shown in the figure) do not appear to be influenced greatly by width of ore, largely because individual stoping rounds usually are drilled to break only one square-set of ground at a time, regardless of the dimensions of the ore body. With caved-stope methods the ore must be wide enough to cave readily, and this usually means that limiting widths must be greater than those at which there is much variation in costs with different widths of ore. Wide ore (or stopes), whatever the method of stoping, obviously makes possible a higher production rate from the mine and thus reduces general and distributive costs. The effect of rate of output upon total unit production costs will be considered later.

Sorting of ore in the stopes adds considerably to the cost per ton if much sorting is required but may result in a reduced cost per unit of metal. Although only rarely is sorting carried to the degree illustrated by the practice at Cold Springs mine, the effect of sorting on the cost per ton of sorted ore and upon the cost per unit of metal is emphasized by the following computations covering practice at that mine.[53]

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[53] Vanderburg, William O., Methods and Costs of Mining Ferberite Ore at Cold Springs Mine, Nederland, Boulder County, Colo.: Bureau of Mines Inf. Circ. 6673, 1932, 15 pp.

Effects of sorting, Cold Springs mine

	Average value per ton
Run-of-mine ore as broken, containing 0.9 percent of WO_3 (WO_3 figured at \$12 per unit)	\$10. 80
Sorted ore containing 5.72 percent of WO_3 (tungstic acid figured at \$12 per unit)	68. 64
Waste rejected in sorting containing 0.025 percent of WO_3 (tungstic acid figured at \$12 per unit)	. 30

Assume: X = the number of tons from which 1 ton of \$68.64 ore can be sorted,
Y = the number of tons of waste thrown away for each ton of \$68.64 ore obtained,

then $68.64+0.30\ Y=10.80\ X$
and $X-Y=1$

$$\begin{array}{r}
10.80\,X-0.30\,Y=68.64 \\
-0.30\,X+0.30\,Y=-.30 \\
\hline
10.50\,X \qquad\quad =68.34
\end{array}$$

Therefore, $X=6.51$ tons of run-of-mine ore broken for every ton of sorted ore sent to mill,
$Y=5.51$ tons of waste discarded for every ton of sorted ore obtained.

The cost of sorting 1 ton of \$68.64 ore from the run-of-mine product is \$1.95. Expressed in another way, the sorting cost per ton of run-of-mine ore is $\frac{1.95}{6.51}$, or \$0.30. The cost of mining 1 ton of run-of-mine ore is \$2.34. The cost of transporting ore from the mine to the mill on contract is \$1 per ton. The milling cost of run-of-mine ore is taken as \$2.85 and the cost of milling the sorted product is \$3.22 per ton.[54] The mill recovery on the unsorted ore is estimated at 80 percent, and the recovery on the sorted ore is 87 percent.

The profit per ton of ore without sorting is shown in the following calculation:

		Per ton
Mill returns, \$10.80×0.80 (recovery)		\$8. 64
Costs:		
Mining	\$2. 34	
Transportation	1. 00	
Milling	2. 85	
Total direct cost		6. 19
Profit per ton of run-of-mine ore without sorting		2. 45

The profit obtained by sorting is shown in the following calculation:

		Per ton of sorted ore
Mill returns, \$68.64×0.87 (recovery)		\$59. 72
Costs:		
Sorting	\$1. 95	
Mining (6.51 tons at \$2.34 per ton)	15. 23	
Transportation	1. 00	
Milling	3. 22	
Total direct cost		21. 40
Profit per ton of sorted ore		38. 32
Equivalent profit on unsorted ore (\$2.45×6.51)		15. 95
Additional profit obtained by sorting		22. 37

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[54] Vanderburg, William O., Methods and Costs of Milling Ferberite Ore at the Wolf Tongue Concentrator, Nederland, Boulder County, Colo.: Bureau of Mines Inf. Circ. 6685, 1933, 13 pp.

Therefore, the additional profit obtained by sorting, as based on 1 ton of run-of-mine ore, is $\frac{22.37}{6.51}$, or \$3.44, and the total profit is $\frac{38.32}{6.51}$, or \$5.89, as against \$2.45 obtained without sorting.

The economical limit of sorting—namely, the amount that can be paid for sorting before the profit will fall below that obtained without sorting—is calculated as follows:

Where \$2.45 = profit per ton of run-of-mine ore without sorting,
\$3.44 = additional profit by sorting,
\$0.30 = cost of sorting per ton of run-of-mine ore $\left(\frac{1.95}{6.51}\right)$
and X = the maximum amount that can be paid for sorting 1 ton of run-of-mine ore.
Then, $2.45 = 3.44 + 0.30 - X$
$X = \$1.29$, or the economical limit which can be paid for sorting 1 ton of run-of-mine ore.

The foregoing analysis reflects the importance of sorting when dealing with narrow veins. It should be mentioned, however, that sorting, to be effective, requires careful supervision, as it may result in actual loss if not properly done.

In the above cost analysis of sorting it is assumed that the cost of mining per ton of run-of-mine ore is the same whether or not sorting is practiced. Actually, however, if the waste obtained from sorting were not utilized for support the cost of mining would be appreciably higher if all the run-of-mine material were sent to the mill, because of the necessity of providing support for the stopes.

Table 39 presents stoping costs in terms of man-hours of labor in stopes per ton of ore mined.

TABLE 39.—*Man-hours per ton, stoping*

I. OPEN STOPING

Mine	Year	Variation of stoping method	Man-hours per ton of ore mined				Tons per man-shift, all stope labor
			Breaking	Hand and power mucking	Other stope labor	Total	
Average 7 mines, Tri-State district.	1927 to 1930	Irregular pillars	0.252	0.226	[1] 0.076	0.554	14.4
Tri-State [2] district	1937	do		[3].167			
Do	1937	do		[4].084			
Do	1937	do		[4].131			
Do	1937	do		[5].080			
No. 8, Southeast Missouri.	1928	do	.122	.319		.441	18.1
Southeast Missouri	1929	do	.162	.233	.034	.429	18.6
Mascot, Tenn	1929	do	.214	.093	.009	.316	28.5
Marquette No. 1, Mich.[6]	1928	Regular pillars	.342	.310	.051	.703	11.4
Mineville, N. Y.[6]	1927	Irregular pillars	.236	.480		.716	11.2
Edwards, N. Y.	1930	Regular and irregular pillars [7]	.456	.304	.072	.832	9.6
Vanadium, Colo	1931	Regular pillars [8]	.300	.900	[9].240	1.440	5.6

[1] Hand tram.
[2] Nicholson, C. W., Improvements in Mining Practice of Tri-State District: Am. Inst. Min. and Met. Eng., Min. Tech., May 1938, 17 pp.
[3] Scraping to hopper.
[4] Scraping into cans.
[5] Scraping to conveyor belts.
[6] Per long ton of 2,240 pounds.
[7] Irregular pillars in strong ground; regular pillars in weaker ground.
[8] Pillars extracted on retreat, using timber and filling; thin ore.
[9] Timbering and filling.

TABLE 39.—*Man-hours per ton, stoping*—Continued

I. OPEN STOPING—continued

Mine	Year	Variation of stoping method	Man-hours per ton of ore mined				Tons per man-shift, all stope labor
			Breaking	Hand and power mucking	Other stope labor	Total	
Granada, Quebec	1932	Overhand stoping pillars, of lean ore and waste.	0. 809	0. 617	0. 249	1. 675	4. 8
Mary, Tenn	1928	Regular pillars, underhand benching.	. 556	. 467		1. 023	7. 8
Burra-Burra, Tenn.	1928	Sublevel; some pillar mining and shrinkage stoping.	. 348	. 258		. 606	13. 2
Menominee No. 1, Mich.[6]	1928	Sublevel stoping	. 353	. 011	. 038	. 402	19. 9
Spring Hill, Mont	1929–30	Sublevel stoping in wide ore; shrinkage in narrow ore.				1. 081	7. 4
Hartley, Kan	1931	Irregular pillars	. 229	[10] . 218 [11] 1. 160 [12] . 198		. 427	18. 7
Presidio, Tex.[13]	1929	do	1. 280	1. 460		2. 740	2. 92
Mt. Isa, Queensland	1936	Sublevel stoping	. 456	. 159	. 098	. 713	11. 2
Iron Mine A[6]	1928	do	. 200			. 200	40. 0
Do	1936	do				. 107	75. 0
Iron Mine B[6]	1928	do	. 180			. 180	44. 4
Iron Mine C [6]	1928	do	. 140			. 140	57. 1
Iron Mine D [6]	1928	do	. 140			. 140	57. 1
Montreal, Wis.[6]	1928	do	. 192	. 024		. 216	37. 0

II. SHRINKAGE STOPING

Mine	Year	Variation of stoping method	Man-hours per ton of ore mined				Tons per man-shift, all stope labor
			Breaking	Mucking	Timbering	Total	
Nevada - Massachusetts, Nev.	1928	On drift pillars	0. 700	0. 383	0. 376	1. 459	5. 5
Harmony, Idaho	1929	On drift pillars (with grizzlies over chutes).	1. 190	. 060		1. 250	6. 4
Hillside, Ill	1929	On drift pillars in wide ore; on stulls in narrow ore.				. 906	8. 8
Daisy, Ill	1929	On stulls				1. 280	6. 2
Cortez, Nev	1929	On drift pillars	. 901	. 475	. 209	1. 585	5. 0
Teck-Hughes, Ontario	1928	do	1. 400		. 117	1. 517	5. 3
Do	1928–29	do				1. 312	6. 1
Kirkland Lake, Ontario	1930	On timbers	. 900	. 480	. 070	1. 450	5. 5
Vipond, Ontario	1929–30	Partly on stulls; partly on pillars.				1. 176	6. 8
Elkoro, Nev	1930	On stulls	. 555		. 818	1. 373	5. 8
Mogollon, N. Mex	1922	On drift pillars				. 740	10. 8
Sylvanite, Ontario	1929	On stulls				1. 700	4. 7
Lake Shore, Ontario	1929	On drift pillars				. 668	12. 0
Engels, Calif	1928	do				. 598	13. 4
Isle Royale	1928	On stulls				. 360	22. 2
Mt. Hope, N. J.[6]	1930	On stope pillars				. 115	69. 5
Marquette No. 4, Mich.[6]	1929	Semishrinkage between stope pillars	. 473	. 126		. 599	13. 3
Verde Central, Ariz	1929–30	On stulls	. 720		. 330	1. 050	7. 6
Porcupine United, Ontario.	1929–30	do	1. 536	. 0650	. 944	3. 130	2. 55
Hog Mountain, Ala	1935–36	On stulls or drift pillars	1. 580	[14] 1. 000	. 180	[15] 2. 760	2. 8

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[6] Per long ton of 2,240 pounds.

[10] Hand.

[11] Power shovel.

[12] Average.

[13] Howbert, Van Dyne, and Bosustow, Richard, Mining Methods and Costs at Presidio Mine of the American Metal Co. of Texas: Trans. Am. Inst. Min. and Met. Eng., 1931, pp. 38–50.

[14] Includes tramming.

[15] Per ton of ore produced in stopes; development ore not included.

TABLE 39.—*Man-hours per ton, stoping*—Continued

III. FORCED CAVING

Mine	Year	Variation of method	Man-hours per ton or ore mined				Tons per man-shift, all stope labor
			Breaking	Mucking and tramming	Timbering	Total	
Alaska-Juneau, Alaska.	1828	Powder drift blasting				0.086	93.0
Climax Molybdenum, Colo.	1929	do				.184	43.4
Bristol, Mich.[6]	1928–29	Transverse shrinkage stopes on drift pillars. Sides of stopes caved by drilling holes and blasting; stope pillars caved.				.155	51.6
Britannia, British Columbia.	1934?	Powder drift blasting	0.084	0.069	0.012	.165	48.5

IV. CUT-AND-FILL STOPING

Mine	Year	Variation of cut-and-fill stoping	Man-hours per ton of ore mined					Tons per man-shift, all stope labor
			Breaking	Mucking	Filling	Timbering	Total	
Block P, Mont	1929	Horizontal; resuing in thin ore.	1.636	1.812		0.508	3.956	2.0
Cold Springs, Colo.	1931	Resuing	[16] 1.240	[16] 0.610	[16] [17] 3.840		[16] 5.690	[16] 1.4
Do	1931	do	[18] .190	[18] .094	[18] .590		[18] .874	[18] 9.1
McIntyre, Ontario	1932	Horizontal on timbers, some shrinkage.	[15] 1.520		[15] 0.140		[15] 1.66	[15] 4.82
Teziutlan, Mexico	1930–31	Mostly horizontal cut-and-fill; some open stoping.	1.068	1.788	1.056	.386	4.298	1.86
Matahambre, Cuba.	1928	Horizontal cut-and-fill, 70%; square-setting, 30%.	.690	.920	0.670		2.280	3.5
Pecos, N. Mex	1928	52% horizontal cut-and-fill; 37% square-sets; 11% shrinkage.	[15] .774	[15] .865	[15] .362	[15] .864	[15] 2.865	[15] 2.79
Do	1929	do	.917	1.015	.454	1.022	3.408	2.34
Champion, Mich	1930	Sublevel inclined	.550	.730	0.210		1.490	[19] 5.39
Hecla, Idaho	1928	Stringer-set	.256	0.451		.611	1.318	6.07
United Verde, Ariz.	1930?	Horizontal	.135	.547	0.466		1.148	6.97
Eagle, Colo	1930?	Horizontal between square-set aisles.	.280	.410	.180	.370	1.240	6.45
Pilares, Mexico	1929	Horizontal and rill					1.909	4.19
Atlanta, Idaho	1934	Horizontal	1.018	.986			2.004	4.00
Park City Consolidated, Utah.	1935	Inclined	.479	0.479		.479	1.437	5.57

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[6] Long tons of 2,240 pounds.

[15] Figures based on ore from stopes only; development ore not included.

[16] Per ton of sorted ore (6.51 tons broken produces 1 ton sorted ore).

[17] Sorting.

[18] Per ton broken.

[19] Includes sorting 40 percent broken as waste.

TABLE 39.—*Man-hours per ton, stoping*—Continued

V. SQUARE-SET STOPING

Mine	Year	Variation of square-set stoping	Man-hours per ton of ore mined					Tons per man-shift, all stope labor
			Breaking	Timbering	Mucking	Filling	Total	
Argonaut, Calif.	1929	Horizontal floors	2.054		0.475	0.347	2.876	2.78
United Verde Extension, Ariz.	1928	Vertical slices 2–3 sets wide and 10–20 sets long.					1.653	4.84
United Verde, Ariz.[20]	1930	Inclined stepped faces	0.383		.872	[9].606	1.861	4.30
Park-Utah, Utah	1928	Rilled sections	.740		.360	[21].830	1.930	4.14
Tintic Standard, Utah.	1929	Vertical sections, 2 sets wide and up to 20 sets long.	.733	0.876	.827	.356	2.792 [21]0.449	2.47
Silver King Coalition, Utah.	1929	Horizontal sections with some stringer set-and-fill.	.934	.683	[22] 2.154		3.771	2.12
Eagle, Colo	1930?	Horizontal floors	.311	.592	.592	.228	1.723	4.64
Bawdwin, Burma	1922?	Rill sections	[23].516		1.95		7.110	1.10

VI. BLOCK CAVING

Mine	Year	Man-hours per ton		Tons per man-shift	
		Stoping and drawing only	Total underground labor	Stoping and drawing only	Total underground labor
Ohio Copper, Utah	1911	0.127	0.469	63.0	17.0
Ray, Ariz	1927	.0787	.748	101.6	10.7
Do	1928	.0692	.661	115.6	12.1
Miami, Ariz	1925–29		.296		27.0
Humboldt, Ariz	1928	.128		62.8	
Braden, Chile	1927		.606		13.2
Do	1928		.656		12.2

VII. SUBLEVEL CAVING

Mine	Year	Man-hours per ton of ore mined					Tons per man-shift, all stope labor
		Breaking	Mucking	Timbering	Other stoping labor	Total	
Eureka-Asteroid, Mich	1929	0.194	0.179	0.038	0.139	[6] 0.550	14.54
Montreal, Wis	1929	.176	.102	.074	.045	[6].397	20.15
Do	1936					.228	+35.0
No. 16	1930					[6].636	12.58
No. 19	1930					[6].200	40.00
No. 20	1930					[6].365	21.92
Old Dominion, Ariz.[24]	1929					[24].462	17.30

[6] Per long ton of 2,240 pounds.
[9] Timbering and filling.
[20] Square-set stopes only.
[21] General stope labor.
[22] Includes considerable sorting.
[23] Timbering and breaking.
[24] Per short ton, Morenci slide (inclined) method.

TABLE 39.—*Man-hours per ton, stoping*—Continued

VIII. TOP-SLICING

Mine	Year	Man-hours per ton of ore mined				Tons per man-shift, all stope labor
		Breaking	Mucking	Timbering	Total	
Mesabi range, Minn.:						
No. 1	1928	0.090	[25] 0.359	0.276	[6] 0.725	[6] 11.03
No. 2	1929				[6].515	[6] 15.52
No. 2	1937				[6].364	[6] 22.00
No. 3	1929				[6].585	[6] 13.68
No. 4	1929				[6].540	[6] 14.80
No. 5	1929				[6].617	[6] 12.96
No. 6	1929				[6].645	[6] 12.40
Vermilion range No. 7, Minn	1929				[6].401	[6] 19.94
Cuyuna range No. 8, Minn	1929				[6].446	[6] 17.94
Marquette range, Mich.:						
No. 9	1929	.093	[25].164	.276	[6].533	[6] 15.01
No. 11	1929				[6].919	[6] 8.70
No. 12	1929				[6].556	[6] 14.39
No. 13	1929				[6].695	[6] 11.50
Coronado, Morenci, Ariz	1917				.714	11.20
Calumet and Arizona, Ariz	1916				.800	10.00
Miami, Ariz	1916				.800	10.00
Bordo, Mexico	1925				2.360	3.38
Judge, Utah	1937	.870	1.460	.420	2.750	2.91

[25] Scraping.
[6] Per long ton of 2,240 pounds.

SELECTION OF STOPING METHOD

Stoping methods and the applicability of each have been discussed and compared in the preceding pages, and the various points there emphasized should be considered in selecting a stoping method for the mining of any particular ore deposit.

Sometimes conditions are such that only one stoping method can be employed because of the limitations imposed by the physical characteristics of the deposit. In other instances there may be a choice of methods or a choice between variations of the same method. Sometimes the best method cannot be determined until one or more stopes have been opened, as the way the ground stands in development openings may be very misleading as a basis for foretelling its action in and around larger stope excavations.

In most mines operated over an extended period of years, mining methods and practices are evolved gradually to best suit conditions that may, in turn, change as the mines become deeper and the workings more extensive. The physical characteristics of the deposit, as indicated in the foregoing discussion, often will definitely eliminate certain methods from consideration, and other factors discussed under Comparison of Stoping Methods will narrow the field of choice further. Additional factors not previously discussed are considered below briefly. Safety, health, and welfare of the workmen obviously are of major importance, and usually will be affected more by details of practice than the stoping method employed.

GRADE OF ORE

The grade of ore may have an important bearing upon the selection of a stoping method. Thus, if the ore is low-grade it may be more profitable to employ a low-cost method even though considerable ore is lost; on the contrary, if the ore is high-grade, a greater profit may result from the use of a higher-cost stoping method to insure complete extraction.

Since the prices of metals fluctuate between rather wide limits, depending on economic conditions, the value of the ores fluctuates also, and ore that it might pay to leave today may become valuable enough later to warrant adoption of methods that will permit its ultimate extraction. Where lower-grade ores occur in proximity to those of higher grade, either around or above them, it may require nice judgment to decide whether it is best to risk the loss of much or all of the lower-grade ore or to adopt methods that will permit immediate mining of the better-grade ore without risk of caving and thus preclude recovery of the lower-grade ore later.

Although the percentage of ores that are beneficiated before shipment for smelting has been increasing for many years, considerable ore is still shipped directly from the mines without treatment. Low-grade ore usually must be concentrated, and under some conditions concentration of fairly high grade ores may be profitable as it eliminates freight on worthless material and often objectional constituents on which a penalty is imposed by the purchaser. If, however, the ore can be kept clean in stoping by preventing dilution from walls and capping it may be more profitable to ship without beneficiation, thus making unnecessary large capital investment for a milling plant and the cost of operating it. Thus, an iron ore may be penalized for silica or pyrite from the walls of the deposit when one mining method (for example, block caving) is used, whereas by employing another method (for example, top slicing) these impurities may be virtually eliminated. In other instances it may be more profitable to employ a low-cost, nonselective mining method, accept considerable dilution in mining, and erect and operate a milling plant.

DISTRIBUTION OF THE ORE MINERALS

The distribution of the ore minerals within the ore body may affect the choice of stoping method. Several types of distribution may be mentioned in this connection—(1) uniform dissemination of ore-mineral particles throughout the deposit; (2) erratic mineralization, with the ore minerals concentrated in lenses or bunches throughout a surrounding mass of low-grade ore or barren rock; (3) bands or irregular and disconnected stringers of the high-grade ore alternating with bands or stringers of waste rock or lean ore; (4) deposits containing two or more grades or classes of ore differing from each other in predominant ore minerals and closely associated; and (5) deposits, the boundaries of which are gradational, that is, where there is no sharp demarcation between ore and walls but the grade of ore decreases gradually toward the margins of the deposit.

1. If the ore minerals are distributed uniformly a flexible, selective stoping method is not required. In large deposits of this type (commonly of low grade), low-cost caved-stope methods are applicable if the physical characteristics of the rock are suitable for caving.

2 and 3. In deposits where the mineralization is erratic (types 2 and 3) there may be a choice between selective and nonselective methods. If selective methods are employed, stoping is confined to the lenses or stringers of higher-grade ore, taking care to keep dilution at a minimum. If nonselective methods are employed, the entire mass is mined and separation of ore minerals from rock is accomplished in an ore-dressing plant or by hand sorting. In general, selective methods are preferable if an ore of good grade can be produced thereby and if the portions of the deposit mined are each comparatively large, few in number, and isolated from each other. Nonselective methods, designed to obtain low mining costs, are especially adapted to large low-grade deposits in which the higher-grade bunches, lenses, or stringers are small. Selective methods include square-set stoping, open stoping in flat or low-dipping beds, and cut-and-fill stoping. Block caving and sublevel caving are nonselective methods, whereas shrinkage stoping, sublevel stoping, and top slicing may be selective under suitable conditions or nonselective under others.

4. In deposits containing two or more grades, classes, or kinds of ore, selective methods are essential if the various ores must be kept separate in mining. Square-set stoping and some forms of open stoping may be employed. Horizontal cut-and-fill stoping may also be applicable if separate chutes are provided for each kind or grade of ore, or with less flexibility if the ore chutes are cleaned out before changing from one ore to another. Shrinkage stoping affords no opportunity for sorting different types of ore in the stopes.

5. In vein deposits having so-called marginal or assay boundaries, especially if the grade of the ore cannot be approximated by visual inspection, sampling must be done after each round is blasted to determine where and how to drill the next round. This is often the case in gold mines and in copper mines where copper sulfides are intimately mixed with larger quantities of iron pyrites. Obviously, stoping methods permitting a high degree of selectivity must be employed under these conditions. Some large massive deposits are enveloped by a gradational zone. If the volume of this zone is small compared to that of the entire mass, nonselective methods may be applicable, since the contaminating gradational material contains some ore mineral and the amount inadvertently broken with the ore will be too small to affect the grade of ore seriously. If, however, the gradational zone is thick, it may be necessary to employ a selective method that at the same time will not leave the mine in such a condition that the marginal ore cannot be mined later when metal prices are more favorable.

CAPITAL REQUIREMENTS

The amount of capital available may restrict the choice of a stoping method. Some methods, especially block caving, require large capital expenditures for general development and stope preparation before production of ore by stoping can be begun. Open stoping and ordinary shrinkage stoping require comparatively small expenditures for stope development, and production can begin almost as soon as development drifts have penetrated the ore. Funds for exploiting a mineral deposit may have to cover not only cost of mine and stope development, plant and equipment, and operating capital for that part of the productive period before any income is received from sales of ore or metal, but also the cost of an ore-dressing plant or even a smelting plant. In order that available funds may be adequate for all capital requirements, it may be necessary to adopt a stoping method that will permit the earliest possible commencement of stoping with a minimum expenditure, even though a lower operating cost, better recovery, and greater over-all profits might be possible were a different method adopted. Methods, such as shrinkage, that tie up a large proportion of the ore broken may result in depletion of capital before production attains a profitable rate if the project is inadequately financed.

COST OF MINE TIMBER

The availability and cost of mine timber may be important factors in some districts; a mining problem that would otherwise be solved easily may become very difficult because of the prohibitive cost of timber. In some instances, cut-and-fill stoping may have to be used because of the high cost of timber, where better recovery of ore and less dilution would result by using square-set stoping. In other instances an abundant and cheap supply of timber, together with a high cost of material for filling, might force the use of square setting even though no other advantage would be gained. The quality of the timber available may affect the selection of stoping method as between square-set stoping and top slicing. In top slicing the stopes remain open only a short time and tough rather than long-life timber is required.

MARKET CONDITIONS

Most metals are subject to wide fluctuations in price and demand; it is therefore advisable to consider the effect of these fluctuations upon the operation of the mine, and a stoping method that can be adapted readily to changes in rate of output with minimum attendant additional expense should be favored, other things being equal.

As previously pointed out, a stoping method may be flexible or inflexible from several standpoints, one of which is the degree to which it may be adapted to changes in rate of production or to complete temporary cessation of operations without excessive special preparation or loss of ore.

Sudden expansion of output is usually impossible with slow methods, such as square setting and cut-and-fill stoping, as increased output with these methods usually requires preparing additional stopes for production. This applies also to ordinary shrinkage stoping, top slicing, and sublevel caving unless development work is kept far ahead of ore extraction. Excessive advance development obviously ties up capital unproductively for some time. With sublevel stoping production usually can be forced considerably by merely increasing the number of benches being worked simultaneously, the number of drills on a bench, or both. With block caving, best results usually can be obtained by a regular rate of drawing, which should not be so rapid as to cause piping through of broken capping or to give too little time for the caved ore to break fine enough to pass the draw raises; moreover, it should not be so slow that excessive weight will be thrown on the grizzly levels before the ore is completely drawn.

With most mining methods, particularly those used in heavy ground, a sudden reduction in rate of output is likely to be more serious in its effects than a sudden increase. Thus, if curtailment must be brought about by reducing the number of active stopes, a number of stopes must be left standing, which probably will result in caving and loss and dilution of ore unless considerable work is done in placing extra timber or introducing large amounts of filling. If output can be curtailed by merely reducing the scale of operations in each stope, the stopes usually can be kept in better condition although, because of scattered operations, haulage and other costs may be increased considerably.

Operations in open-stope mines usually can be curtailed or stopped entirely without serious difficulty or much preparation, as the walls and ore are strong and firm and may stand for years without much sloughing.

Cessation of drawing in block-caving operations before the block has been drawn completely may result in packing of the broken ore; extraction openings may take weight and crush also. With sublevel caving and top slicing, it is usually necessary to complete active slices before shutting down or at least to pull back to a straight cave line if loss of ore and dilution are to be avoided when operations are resumed later.

FACTORS AFFECTING RELATED MINING OPERATIONS

Stoping is closely related to other underground operations—development, hauling and hoisting of ore and waste, procuring and handling of filling material, and handling of timber, drill steel, explosives, and other supplies.

Thus, the stoping method to be employed should be considered when the general plan of development is laid out and a level interval and the details of development are being selected.

A single longitudinal extraction drift suffices for shrinkage stoping in a vein of moderate width, whereas with cut-and-fill stoping an additional hanging-wall drift may be desirable for the introduction of fill if the dip of the vein is flat. Shrinkage stopes usually require a series of closely spaced chutes along the haulage drifts, whereas chutes for cut-and-fill stopes may be spaced considerably farther apart; a series of raises from the stope often is essential for introduction of filling in cut-and-fill and square-set stopes, whereas only an occasional raise for ventilation, access to the stope, and lowering equipment and supplies is needed for shrinkage stopes.

Sublevel caving, sublevel stoping, and top slicing require considerable sublevel and raise development, the amount depending to some extent on the details of stoping practice as well as on the size, shape, and dip of the ore body.

The haulage-level interval and stoping method are related, although the level interval may be controlled principally by the degree of continuity of the ore bodies and the requirements for exploratory workings. If the levels are close together the available stoping height obviously is reduced, and if stopes are silled above the level on drift pillars and a floor pillar is left also under the level above, the actual stoping height will be very small. On the other hand, a short level interval may permit the use of a less expensive stoping method. Thus, shrinkage stoping might be used with a short interval, whereas under the same conditions a long level interval might require use of cut-and-fill stoping or square-set stoping because, with the short interval, the stope could be completed quickly before serious sloughing begins, whereas with a long interval sloughing and perhaps collapse of the stope might take place before a shrinkage stope could be carried through to the level above and the ore drawn out.

Development for top slicing and sublevel caving varies with the method of handling ore in the slices and the arrangement of the slice drifts, and variations in details of sublevel caving practice affect the number of haulage drifts and crosscuts, main passes, and branch raises.

HAULAGE AND HOISTING

Up to a certain point, concentration of stoping operations within a comparatively small area makes for economy in operation, but if carried to extremes the haulageways may become congested. Some methods, such as forced caving, block caving, and sublevel stoping, make possible a high rate of output from individual stopes. Often entire trains can be loaded from one chute or one stope, whereas with other mining methods where output from individual stopes is more restricted haulage costs may be increased considerably by the necessity for switching and moving from one stope to another to fill a train of cars.

When tramming distances are short and only a small tonnage is available from each chute or stope, hand tramming actually may be cheaper than locomotive haulage; but where hauls are long and large tonnages are drawn from each stope, mechanical haulage is most economical. With some methods, ore must be drawn frequently but in small amounts from each chute, as in shrinkage stoping while the stope is advancing; even when the stope has been completed and during final drawing it may be necessary to draw the ore, a little at a time, to permit scaling or trimming of the walls, cleaning broken ore from flat spots in the footwall, and timbering. Unless a transfer drift with scraper or cars is employed, it is necessary to draw frequently from chutes in top slicing and sublevel caving when slicing approaches the level, so as to keep them empty and avoid retarding the work of the miners in the slices.

Where filling must be obtained from sources outside the stopes, it may be necessary to lay out the haulageways with a view to avoiding interference with hauling of ore. If the stoping method requires the use of large quantities of timber, special timber shafts or a special compartment in the main shaft to accommodate a large cage may have to be provided.

HEALTH AND SAFETY

The chief single cause of serious accidents in underground mines has been falls of ground. This type of accident can be prevented only if the stoping method is such that adequate support and protection can be provided. Safe practices, education of the miners, and safety rules and regulations are essential, of course, but these avail little if the stoping method is unsuited to the ground.

The health of miners depends, among other things, on maintenance of comfortable temperatures in the working places, elimination of dust, and the supplying of plenty of fresh air, for all of which good ventilation of the working places is necessary. Adequate ventilation is more difficult of achievement with some stoping methods than with others, but with modern equipment and proper attention it can be maintained with any of them. Large open stopes are easily ventilated; shrinkage, cut-and-fill, and square-set stopes require enough openings at top or bottom, or both, to insure circulation, and in addition fans may be required. Slices in top-slicing and sublevel-caving operations probably are the most difficult to ventilate, and it often is necessary to supply each working face independently with fresh air by means of injectors, blowers, or fans.

DIRECT COST OF UNDERGROUND MINING

Costs of exploration, development, haulage and hoisting, and stoping have been discussed separately in the preceding pages. Tables 40 and 41 present data on total direct mining costs. In table 40 costs are distributed between exploration and development, stoping, haulage and hoisting, pumping, general underground expense, and general surface expense applicable to mining. In table 41, the distribution is made as between labor and supervision, compressed air, rock drills and drill steel, power, explosives, timber, and other supplies.

The tabulated costs cover all underground operations, proportion of surface expense directly applicable to mining, mine office, engineering, sampling and assaying, and supervision. They do not include taxes, depreciation, depletion, interest, marketing, and home office expense, which in the aggregate are ordinarily about 25 percent and may be as much as 50 percent or more of the direct cost.

Although most of the data are taken from figures prepared in accordance with the Bureau of Mines outline for mining-method reports it is realized that, due to differences in accounting practices at different mines and because different authors prepared the figures, they are not strictly comparable. Thus, if costs were compiled in strict accordance with the outline, the power for compressing air would be included under the column "Compressed air, drills, and steel," and only other power under the column "Power." It is obvious in the tables that for many of the mines power for air compression has been included under "Power." However, it is believed that the data here given are useful as a basis upon which to estimate probable direct mining costs at a new mine, and, together with cost data presented in previous tables, should be helpful to the mine superintendent in analyzing his own costs.

TABLE 40.—*Total direct mining costs at underground mines*

Mine and year	Principal stoping method	Rate of output per month or year, tons	Base wage rate		Direct mining costs per ton of ore mined [1]						
			Miners (machinemen)	Muckers	Exploration and development	Stoping	Haulage and hoisting	Pumping	General underground	General surface [2]	Total
Mineville, N. Y., 1927 [3]	Open stoping with pillars	692,726, year	[4] $2.88	[4] $2.56	$0.103	$0.416	$0.705	$0.026	----------	----------	$1.250
Tri-State No. 1, 1928	do	11,460, month	4.25	[5] 3.50	.049	.532	.217		$0.187		.985
Tri-State No. 2, 1928	do	9,730, month	4.25	3.50	.508		.200	.165		$0.120	.993
Tri-State No. 3, 1927	do	132,384, year	4.25	3.50	.480		.165	.162		.194	1.001
Waco, Mo., 1927–28	do	182,140, average per year.	3.50	3.00–3.25	.540		.155	.036	.021	.033	.785
Barr, Kans., 1929	do	11,335, month	4.75	4.00	.000	.544	.217	.069		.032	.862
Hartley-Grantham, Kans., 1929	do	10,000 month	4.25	3.75	.000	.516	.196	.049		.021	.782
Hartley, Kans., 1932	do	30,000 month	4.75	4.00	.000	.572	.152	.063		.041	.828
No. 8, SE. Missouri, 1928	do	168, 089, year	5.05	5.00	.045	.483	.222	.031	.105	?	.886
Marquette No. 1, Mich., 1928 [3]	do	420,000, year	Contract [1]		.218	.770	.244	.180		?	1.412
Burra-Burra, Tenn., 1928	Sublevel stoping	473,292, year	[6] 3.92 [7] 7.17	[6] 3.60 [7] 4.97	[8].229	.332	.386	.145		.043	1.135
Marquette No. 2, Mich., 1928 [3]	do	120,000 year	Contract		.251	.417	.149	.193		.066	1.076
Menominee No. 1, Mich., 1928	do	200,000 year	Contract		.211	.467	.173	.052		.027	.930
Spring Hill, Mont., 1929–30	Modified sublevel stoping	4,814, month	5.50	5.00	[9].377	1.072	.131	.218		.147	1.945
Mt. Isa, Queensland, 1935–36	Sublevel stoping	703,924, 1 year	3.76	3.36	[10].103	1.211	.271	.143		.050	1.778
Mascot, Tenn., 1929	Open stoping with pillars	52,860, month	Contract		.053	.222	.278	.086		(?)	.639
Hanover, N. Mex., Bessemer, 1930.[3]	Open stoping, underhand benches.	(?)	(?)	(?)	(?)	.343	.187	(?)	(?)	(?)	(?)
El Potosi, Mexico, 1933	do	53,500, 1 month	Contract		.202	.337	.105	.002	.060	.143	.849
Mary, Tenn., 1928	do	108,519, year	Contract		.355	.784	.475	.154		.060	1.828
Isabella, Tenn., 1928	do	74,000, year	Contract		.302	.391	.246	.076		.015	1.030
Vanadium, Colo., 1931	Open stoping, with pillars	4,875, month	(?)	(?)	.200	1.350	.240	.090		.080	1.960
Presidio, Tex., 1929	do	54,524, year	4.00	3.00	1.065	2.276	.533	(11)	(11)	.127	4.001
Granada, Quebec, 1932	Open stoping, overhand	3,104, 1 month	5.04–5.54	4.24–4.74	1.163	2.765	.974	.111		(?)	5.013
Teziutlan, Mexico, 1931	Open stoping and cut-and-fill stoping.	65,764, year	(?)	(?)	.926	1.571	.269	.312		.160	3.238
Grand Republic, Colo., 1935	Open stoping	28,515, year	4.00	3.20	1.283		.085	.052		.498	1.918
Summitville, Colo., 1937	Open stoping and shrinkage	3.250 per month	4.25	3.50	1.381	2.036	.375	(11)	(11)	(11)	3.792
Harmony, Idaho, 1929	Shrinkage	3,800, month	6.00	6.00	.650	2.050	.210	----------	.230	----------	3.140
Nevada-Massachusetts, Nev., 1928.	do	40,924, year [12]	5.25	4.75	.907	2.743	.190	.105	(?)	(?)	3.945
Hillside, Ill., 1929	Shrinkage, some square-set	51,208, year	4.00–4.50	3.50	.383	1.276	.399	.404		.138	2.600
Daisy, Ill., 1929	Shrinkage and stulled stopes	42,000, year	4.50–5.00	4.00	1.165	1.395	.816	.340		.150	3.866
Cortez, Nev., 1929	Shrinkage	43,806, year	5.75	5.25	1.427	1.972	.467	(?)	(?)	(?)	3.866
Eighty-Five, N. Mex., 1930	do	7,220, 1 month	4.10–5.00	3.60	1.00	1.970	1.140	.520	.790	.070	5.490
Verde Central, Ariz., 1929–1930.	do	10,000 per month	5.69	5.06	(?)	[13] 1.661	.580	.111		.033	-------

TABLE 40.—*Total direct mining costs at underground mines*—Continued

Mine and year	Principal stoping method	Rate of output per month or year, tons	Base wage rate		Direct mining costs per ton of ore mined						
			Miners (machinemen)	Muckers	Exploration and development	Stoping	Haulage and hoisting	Pumping	General underground	General surface	Total
Hecla, Idaho, 1928	Stringer-set-and-fill	312,942, year	$3.75 plus bonus	$3.25	$0.638	$1.750	$0.322	$0.227		$0.360	$3.297
Morning, Idaho, 1928	do	276,890, year	5.00	4.50	.442	2.371	.530	.698		.204	4.245
Silver King Coalition, Utah, 1929.	Stringer-set-and-fill, and square set.	180,208, year	4.50	4.25	2.086	3.502	1.848	.309	(11)	(11)	7.745
Colorada, Mexico, 1929	Cut-and-fill, square set, and shrinkage.	43,713, per month, average.	Contract		.561	1.203	.166	.278		.233	2.441
Capote, Mexico, 1929	Undercut caving, top slicing, cut-and-fill.	23,700 per month	Contract		.057	.894	.202	.083		.242	1.478
Veta Grande, Mexico, 1929	Square set, top slice, cut-and-fill.	7,100 per month	Contract		.322	1.529	.108	.122		.115	2.196
Magma, Ariz., 1928	Cut-and-fill and square set	263,094, year	4.95	4.13	.736	2.849	.616	1.166		.130	5.497
McIntyre, Ontario, 1931	Cut-and-fill and shrinkage	557,104, year	4.80	4.24	.655	1.489	.135	.660		.134	3.073
Park City Consolidated, Utah, 1935.	Cut-and-fill and open stulled stopes.	6,219 per month, average.	5.25	4.75	1.010	1.225	.747	.201	.055	.175	3.413
Hollinger, Ont., 1934	Cut-and-fill, shrinkage	1,900,490, year	4.80	4.24	.798	1.442	.393	.025	.220	.095	2.973
Ground Hog, N. Mex., 1930	Cut-and-fill and square set	3,715, per month	4.25–5.00	3.35–4.00	1.393	1.762	.628	.080		.127	3.990
Golden Anchor, Idaho, 1937	Resuing	9,547, year	5.50	5.00	3.150	3.78	(11)	(11)	.06	(11)	6.99
Atlanta, Idaho, 1934	Cut-and-fill, square set	6,174, 1 month	5.00	4.50	.272	2.577	.242		0.159		3.250
Argonaut, Calif., 1929	Square set	7,800, 1 month	4.50	4.00	.165	2.439	.519	.577		.291	3.991
United Verde Extension, Ariz., 1928.	do	275, 212, year	(?)	(?)	.611	1.959	.508	.039	.435	.738	4.290
Bunker Hill & Sullivan, Idaho, 1928.	do	452,345, year	(?)	(?)	.053	2.272	.381	.996		.195	[26] 3.897
Bunker Hill & Sullivan, Idaho, 1931.	do	460,366, year	(?)	(?)	.066	2.476	.308	.604			3.454
Page, Idaho, 1928	do	90,460, year	(?)	(?)	1.170	2.150	.160	.180		.160	3.820
Park-Utah, Utah, 1928	Square set, cut-and-fill	192,250, year	5.25	4.75	.828	2.426	.282	.583		.543	4.662
Tintic Standard, Utah, 1929	Square set	8,758, 1 month	4.75	4.25	1.143	4.973	1.158	.322		1.888	9.484
Bawdwin, Burma	do	14,643 [3]	Bonus system		.580	1.793 [27] .440	.761	.099		[28] .018	3.691
Gold Hill, Idaho, 1936	do	3,579, 1 month	4.50	4.00	[29] .370	[30] 2.320	.590	.750		.420	4.450
Eureka Standard, Utah, 1933	do	3,737, 1 month	3.50	3.00	.960	3.230	.660	.540	2.010		7.400
Victoria, British Columbia, 1934?	do	133,195, year	Bonus system		.131	1.131	.134	[31] .212		.160	1.768
Dayton Consolidated, 1936	do	43,180, year	----------	----------	1.390	1.672	.452	.356			3.870
Ray, Ariz., 1928	Block caving	3,243,159, year	Contract and bonus		----------	.3468	.1106	.0573		.1619	.6766
Inspiration, Ariz., 1928	do	4,879,646, year	Contract system		.211	.208	.144	.012		.008	.583
Miami, Ariz., 1925–1929	do	4,139,000 per year	$5.06+ 45 percent on contract.		[32] .1000	.1363	.0863	.0570		.0198	.3994

Teck-Hughes, Ontario, 1928	do	29,810, 1 month	4. 75	4. 25	[14] . 433	1. 557	. 443	.202		. 435	3. 070
Kirkland Lake, Ont., 1928	do	4,108, 1 month	4. 75	4. 25	2. 870	1. 620	1. 610	.140		. 520	6. 760
Elkoro, Nev., 1930	Shrinkage and cut-and-fill	57,539, year	(?)	(?)	1. 290	1. 610	. 416	.440		. 257	4. 013
Vipond, Ont., 1929–30	Shrinkage	104,381, year	4. 80	4. 24	. 728	1. 275	. 629	.148		(11)	2. 780
Porcupine United, Ont., 1929–30	do	600 per month average	4. 80	4. 24	. 202	2. 923	2. 301	.842		. 028	6. 296
Engels, Calif., 1928	do	303,301, year	5. 00	4. 50	. 479	. 760	. 438	(11)		----------	1. 677
Marquette, Mich., No. 4, 1929 [3]	Semishrinkage	117,224, year	Contract		. 237	. 897	[15] . 229	.059		. 010	1. 432
Mt. Hope, N. J., 1930 [3]	Shrinkage	176,474, year	4. 68–6. 65 (contract)	4.68 plus bonus	. 104	[16] . 632	. 459	.111		. 113	1. 419
Eustis, Quebec, 1934	do	71,709, year	(?)	(?)	. 651	[17] . 435	[17]. 347	.440		[18] . 224	2. 097
Hog Mountain, Ala., 1935–36	do	3,400, per month [19]	3. 25 (9 hours)	2. 50	. 750	1. 154	. 245	.205		----------	[19] 2. 354
Alaska-Juneau, Alaska, 1928	Forced caving	3,670,910, year	5. 50 plus bonus.	5. 00	. 061	. 122	. 114	----------	----------	----------	. 297
Beatson, Alaska, 1922–1926	do	336,400 per year	4. 50–5. 40	4. 00–4. 75	. 053	. 366 [20] . 237	. 137	----------	----------	----------	.793
Britannia East Bluff, B. C., 1934?	do	398,644, year	Bonus system		. 080	. 235	. 060	.069		. 070	. 514
Cold Springs, Colo., 1931	Resuing (cut and fill)	141 per month [21]	4. 00	4. 00	[22] 5. 635	[22] 5. 951	[22] 2. 143	[22] 3. 442			[22]17.171
		9,154 per month [23]	----------	----------	[24] . 865	[24] 914	[24] . 329	[24] . 529			[22] 2. 637
Lucky Tiger, Mexico, 1924	do	6,080, 1 month	Contract		3. 100	6. 313	1. 617	. 320	(11)	(11)	11. 350
Matahambre, Cuba, 1928	Cut-and-fill	364,746, year	2. 50 or contract	2. 00	. 711	1. 267	. 388	. 184		. 019	2. 569
Teziutlan, N. Mex., 1930–31	Cut-and-fill and open stoping	65,764, year	(?)	(?)	. 926	1. 571	2. 69	. 312		. 160	3. 238
Pecos, N. Mex., 1927–29	Cut-and-fill, shrinkage, square set.	194,000 per year, average.	4. 50	3. 50	1. 005	2. 278	. 663	. 423		. 010	4. 379
Pilares, Mexico, 1929	Cut-and-fill	696,994 hoisted; 835,036 broken.	Contract		. 399	1. 370	. 370	. 250		. 211	2. 600
United Eastern, Ariz., 1917–25	do	8,230 per month, average.	(?)	(?)	. 518	2. 923	. 275	. 052	. 635		4. 403
Champion, Mich., 1930	Cut-and-fill (sublevel, inclined).	38,700 per month, average.	Contract		. 489	1. 162	. 552	. 133	. 074	. 112	[25] 2. 522

[1] Per ton of ore taken out of mine unless otherwise noted.

[2] Portion of surface expense chargeable to mining operations.

[3] Costs given per long ton (2,240 pounds).

[4] Contract earnings considerably higher.

[5] Trammers (muckers on contract).

[6] Company rate.

[7] Average earnings on contract.

[8] Abnormal development.

[9] Less than average development.

[10] Capital development plus development redemption $0.10 additional.

[11] Distributed over other costs.

[12] About 15,000 tons more ore broken but not trammed and hoisted.

[13] Includes all development in ore body after stoping has begun but not major development and prospecting work

[14] Cost of sinking large deep shaft not included.

[15] About 35 percent of ore broken left in stopes, that is, tonnage broken was about 50 percent more than tons hoisted.

[16] Includes stope development, $0.298.

[17] Redistributed from original data allocating half of "mucking and tramming" to "haulage and hoisting."

[18] Includes haulage to mill, $0.1125.

[19] 25 percent more ore broken than was hoisted.

[20] Stope preparation.

[21] Tons of sorted ore; 6.51 tons broken per ton of sorted ore.

[22] Per ton of sorted ore.

[23] Total tons broken.

[24] Per ton of ore broken.

[25] Per ton of sorted ore; 40 percent of tonnage broken sorted out in stopes as waste.

Braden, Chile, 1928	do	5,174,017, year	Mostly contract		([33])	[34].2793	.0489		.0778	[35].4060
Eureka-Asteroid, Mich., 1929	Sublevel caving	401,680, year [3]	Mostly contract		.317	[36].677	.408	.135	([37])	1.537
Montreal, Wis., 1928	do	450,000, year [3]	Mostly contract		.254	[38].584	.344	[39] .087	.070	1.339
Mesabi No. 1, Minn., 1928	Top slicing	209,300, year [3]	Mostly contract			.729	.128	.250	.058	1.165
Marquette No. 5, Mich., 1929	do	555,919 year [3]	Mostly contract		.052	.739	.210	.090	.004	1.095
Marquette No. 2, Mich., 1928	do	240,000, year [3]	Mostly contract		.251	.707	.149	.193	.066	1.366
Judge, Utah 1937	do	6,540 per month	$5.75	$5.25	1.338	4.199	.551	1.090	.652	7.830
Miami, Ariz., 1916	do				.140	.550	.095	.065	.030	.880

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[3] Long tons of 2,240 pounds.

[11] Distributed over other costs.

[26] Total same as in original summary; costs redistributed from detail cost figures.

[27] Repairs and maintenance.

[28] Loading on railway cars.

[29] Exploration only.

[30] Includes development.

[31] Includes hoisting.

[32] Includes $0.0299 preparatory work above grizzly level.

[33] Included with stoping.

[34] Includes undercutting, hand tramming, and development.

[35] $0.0777 in addition chargeable to head office, etc.

[35] Long tons.

[36] Includes $0.279 general timbering.

[37] Included in other items.

[38] Includes $0.224 general timbering.

[39] Pumping, ventilation, and miscellaneous.

TABLE 41.—*Total direct mining costs at underground mines*

Mine and year	Principal stoping method	Rate of output, per month or year, tons	Base wage rate		Direct mining cost per ton of ore mined [1]						
			Miners	Muckers	Labor and supervision	Compressed air drills and steel	Power	Explosives	Timber	Other supplies	Total
Mineville, N. Y., 1927	Open stoping with pillars	692,726,[2] year	[3] $2.88	[3] $2.56	$0.755	$0.155	$0.087	$0.143	----------	$0.110	$1.250
Tri-State No. 1, 1928	do	11,460, month	4.25	[4] 3.50	.624	.127	.041	.110	----------	.083	.985
Tri-State No. 2, 1928	do	9,730, month	4.25	[4] 3.50	.576	.114	.076	.157	----------	.070	.993
Tri-State No. 3, 1927	do	132,384, year	4.25	[4] 3.50	.675	.075	.102	.117	----------	.032	1.001
Waco, Mo., 1927–28	do	182,140, average per year.	3.50	3.00–3.25	.536	.109		.072	----------	.068	.785
Barr, Kans., 1929	do	11,335, month	4.75	4.00	.529	.066	.048	.103	----------	.116	.861
Hartley-Grantham, Kans., 1929.	do	10,000 per month	4.25	3.75	.535	.052	.037	.107	----------	.051	.782
No. 8, Southeast Missouri, 1928	do	168,089, year	5.05	5.00	.670	(5)	.053	.072	----------	.091	.886
Marquette No. 1, Mich., 1928	do	420,000,[2] year	Contract system		.893	.153	.108	.136	$0.013	.109	1.412
Burra-Burra, Tenn., 1928	Sublevel stoping	473,292, year	[6] 3.92 [7] 7.17	[6] 3.60 [7] 4.97	.744	.088	.035	.126	.024	.118	1.135
Marquette No. 2, Mich., 1928	do	120,000,[2] year	Contract system		.591	.077	.141	.068	.045	.154	1.076
Menominee No. 1, Mich., 1928	do	200,000,[2] year	Contract system		.562	.111	.070	.129	.019	.039	.930
Spring Hill, Mont., 1929–30	Modified sublevel stoping	4,814 per month	5.50	5.00	1.415	.183	.003	.267	.010	.067	1.945
Mt. Isa, Queensland, 1935–36	Sublevel stoping	703,924, 1 year	3.76	3.36	1.046	.197	.117	.234	.045	.139	[8] 1.778
Mascot, Tenn., 1929	Open stoping with pillars	52,860 per month	Contract system		.454	.047	.016	.060	----------	.062	.639
El Potosi, Mexico, 1933	Open stoping with underhand benches.	53,500, 1 month	Contract system		.357	.035	.148	.082	.013	.071	.706 [9] .143
Mary, Tenn., 1928	do	108,519, year	Contract system		1.169	.231	.216	.142	----------	.070	1.828
Isabella, Tenn., 1928	do	74,000, year	Contract system		.675	.090	.051	.111	----------	.103	1.030
Vanadium, Colo., 1931	Open stoping with pillars	4,875, per month	(?)	(?)	1.290	.100	.150	.240	.060	.120	1.960
Presidio, Tex., 1929	do	54,524, year	4.00	3.00	2.859	.597	.049	.294	.025	.177	4.001
Granada, Quebec, 1932	Open stoping, overhand	3,104, 1 month	5.04–5.54	4.24–4.74	2.983	.730	.076	1.024	.059	.142	5.014
Teziutlan, Mexico, 1931	Open stoping and cut-and-fill stoping.	65,764, year	(?)	(?)	1.912	.387	.086	.335	.175	.343	3.238
Golden Messenger, Mont., 1936.	Open stoping, overhand and underhand.	3,750, 1 month	4.50	4.25	1.595	.197	.085	.315	----------	.219	2.411
Grand Republic, Colo., 1935	Open stoping	28,515, year	4.00	3.20	1.352	.050	.126	.288	----------	.102	1.918
Summitville, Colo., 1937	Open stoping and shrinkage	3,250, per month	4.25	3.50	1.942	1.062		.399	.119	.270	3.792
Harmony, Idaho, 1929	Shrinkage	3,800 per month	6.00	6.00	1.670	.300	.220	.360	.230	.360	3.140
Nevada-Massachusetts, Nev., 1928.	do	40,924, year [10]	5.25	4.75	2.398	.125	.301	.455	.182	.484	3.945
Hillside, Ill., 1929	Shrinkage, some square setting.	51,208, year	4.00–4.50	3.50	1.824	.144	.138	.301	.082	.111	2.600
Daisy, Ill., 1929	Shrinkage and stulled stopes	42,000, year	4.50–5.00	4.00	2.679	.352	.140	.370	.120	.205	3.866
Cortez, Nev., 1929	Shrinkage	43,806, year	5.75	5.25	2.445	.430	.116	.495	.090	.290	3.866

Eighty-Five, N. Mex., 1930	do	7,220, 1 month	4.10–5.00	3.60	[11] 2.470	(5)	[11] 1.100	[11].280	[11].270	[11].370	[11] 4,490 [12] 1,000 5.490
Teck-Hughes, Ont., 1928	do	29,810, 1 month	4.75	4.25	2.021	.152	.284	.407	.047	.159	3.070
Kirkland Lake, Ont., 1930	do	4,108, 1 month	4.75	4.25	4.160	.610	.390	.850	.450	.300	6.760
Elkoro, Nev., 1930	Shrinkage and cut-and-fill	57,539, year	?	?	[11] 1.762 [12].834 2.596	[11].048 [12].129 .177	[11].139 [12].025 .164	[11].107 [12].227 .334	[11].164 [12].044 .208	[11].390 [12].144 .534	[11] 2.610 [12] 1.403 4.013
Vipond, Ont., 1929–30	Shrinkage	104,381, year	4.80	4.24	1.703	.378	.091	.331	.046	.231	2.780
Porcupine-United, Ont., 1929–30.	do	600 per month	4.80	4.24	3.971	.769	.899	.330	.120	.207	6.296
Engels, Calif., 1928	do	303,301, year	5.00	4.50	1.098	.189		.267	.032	.091	1.677
Marquette No. 4., Mich., 1929	Semishrinkage	117,224, year [2]	Contract		.862	.179	.039	.148	.014	.190	1.432
Mt. Hope, N. J., 1930	Shrinkage	176,474 year [2]	[13] 4.68–6.65	4.68 plus bonus	.672	.227	.165	.141	.027	.187	1.419
Eustis, Que., 1934	do	71,709, year	?		1.251	(5)	.095	----	----	[14].751	2.097
Alaska-Juneau, Alaska, 1928	Forced caving	3,670,910, year	5.50 plus bonus	5.00	.1803	(5)	.0089	.0594	----	.0480	.2966
Beatson, Alaska, 1922–26 [15]	do	336,400 per year	4.50–5.40	4.00–4.75	.3470	.1187	----	.1402	----	.1871	.793
Britannia East Bluff, British Columbia, 1934?	do	398,664, year	Bonus system		.267	.019	.035	.119	.031	.043	.514

[1] Per ton of ore taken out of the mine, unless otherwise noted.
[2] Long tons of 2,240 pounds.
[3] Contract earnings considerably higher.
[4] Trammers; muckers on contract.
[5] Included in other items.
[6] Company rate.
[7] Average earnings on contract.
[8] Capital development plus development redemption $0.10 additional.
[9] General surface expense.
[10] About 15,000 tons more ore broken than was hoisted.
[11] Development work not included.
[12] Development.
[13] On contract.
[14] All supplies and miscellaneous items.
[15] Distribution estimated by authors from original data.

TABLE 41.—*Total direct mining costs at underground mines*—Continued

Mine and year	Principal stoping method	Rate of output, per month or year, tons	Base wage rate		Direct mining cost per ton of ore mined [1]						
			Miners	Muckers	Labor and supervision	Compressed air drills and steel	Power	Explosives	Timber	Other supplies	Total
Cold Springs, Colo., 1931	Resuing (cut-and-fill)	141 per month [16]	$4.00	$4.00	[17] $10.826	[17] $1.555	[17] $1.848	[17] $.806	[17] $1.172	[17] $0.965	[17] $17.172
		9,154 per month [18]			[19] 1.663	[19] .239	[19] .284	[19] .124	[19] .179	[19] .148	[19] 2.637
Lucky Tiger, Mexico, 1924	Mostly resuing	6,080, one month	Contract		6.837	1.200	.711	.909	.956	.737	[17] 11.350
Matahambre, Cuba, 1928	Cut-and-fill and some square set.	364,746, year	2.50 or contract	2.00 or contract	1.588	.161	.087	.200	.269	.264	2.569
Teziutlan, Mexico, 1930–31	Cut-and-fill and some open stoping.	65,764, year	?	?	1.912	.387	.086	.335	.175	.343	3.238
Pecos, N. Mex., 1929 only	Cut-and-fill, shrinkage, square set.	217,121	4.50	3.50	3.308	.141	.091	.259	.422	.677	4.898
Pilares, Mexico, 1929	Cut-and-fill	696,994, [20] 835,036, [21] year.	Contract		1.593	.231	.055	.232	.319	.170	2.600
United Eastern, Ariz., 1917–25	do	8,230 per month	(?)	(?)	2.628	.208	.289	.513	.462	.303	4.403
Champion, Mich., 1930	Sublevel inclined cut-and-fill	38,700 per month	Contract		1.751	.197	.192	.172	.109	.101	2.522
Hecla, Idaho, 1928	Stringer-set-and-fill	312,942, year	3.75 plus bonus	3.25 plus bonus	1.868	.070	[22] .145	.112	.357	.317	[23] 2.869 [24] .428 3.297
Morning, Idaho, 1928	do	276,890, year	5.00	4.50	3.088	.122	.211	.158	.417	.249	4.245
Silver King Coalition, Utah, 1929.	Stringer-set-and-fill and square set.	180,208, year	4.50	4.25	5.418	.854	.370	.486	.340	.277	7.745
Colorada, Mexico, 1929	Cut-and-fill, square set, and shrinkage.	43,713, per month	Contract		1.273	.235	.070	.278	.257	.328	2.441
Capote, Mexico, 1929	Undercut caving, top slice, cut-and-fill.	23,700, per month	Contract		.657	.111	.194	.108	.181	.227	1.478
Veta Grande, Mexico, 1929	Square set, top slice, cut-and-fill.	7,100, per month	Contract		1.139	.157	.066	.175	.441	.218	2.196
Park City Consolidated, Utah, 1935.	Cut-and-fill and open stulled stopes.	6,219, per month	5.25	4.75	2.581	.137	.219	.154	.253	.069	3.413
Magma, Ariz., 1928	Cut-and-fill and square set	263,094, year	4.95	4.13	3.654	.326	.197	.209	.532	.579	5.497
McIntyre, Ont., 1931	Cut and fill and shrinkage	557,104, year	4.80	4.24	1.938	.052	[22] .176	.309	.240	.358	3.073
Hollinger, Ont., 1934	do	1,900,490, year	4.80	4.24	1.898	.251	.062	.245	.193	.324	2.973
Ground Hog, N. Mex., 1930	Cut-and-fill and square set	3,715, month	4.25–5.00	3.35–4.00	2.793	.169	.179	.187	.318	.344	3.990
Golden Anchor, Idaho, 1937	Resuing	9,547, year	5.50	5.00	[24] 5.120		.580	.440	.110	.740	6.990
Atlanta, Idaho, 1934	Cut-and-fill, and square set	6,174, 1 month	5.00	4.50	2.325	.163	.053	.345	.304	.060	3.250

For purposes of direct comparison it is regretted that the tabulated figures do not show for each mine whether development was being conducted at a normal, subnormal, or abnormal rate. Normal rate may be taken as that at which new ore is developed at approximately the mining rate, thus maintaining a constant ore reserve tonnage. In most instances the ratio of tons of ore mined to tons broken is not shown, and the figures fail to reveal the percentage of total rock broken that was sorted out as waste. These factors, together with variations in wage rates and unit costs of supplies, may have considerable influence on the costs in individual instances. The production rate also affects the direct unit costs, particularly those for pumping, transportation, general underground and surface charges, and supervision.

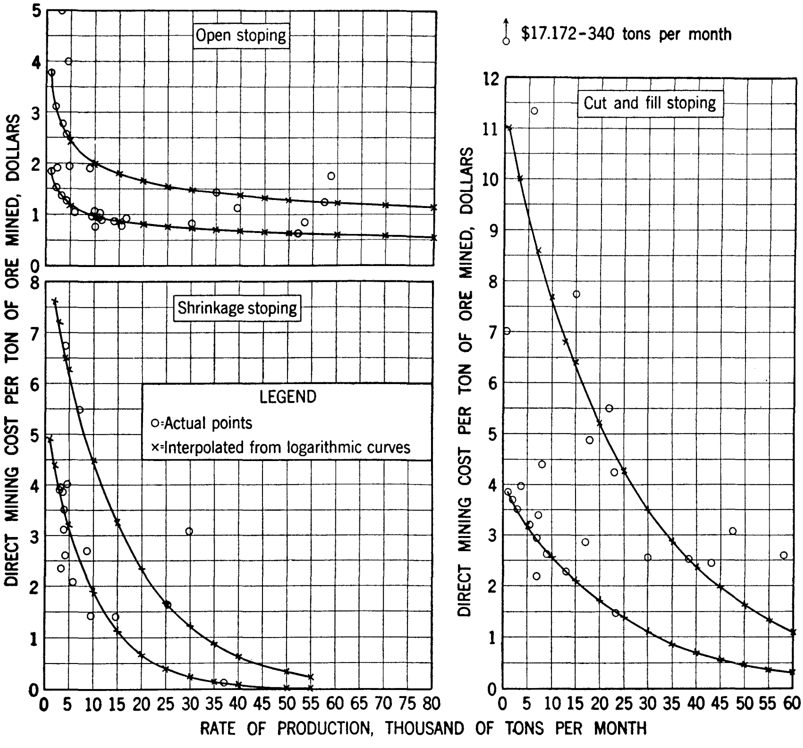

FIGURE 120.—Curves showing effect of production rate on total direct mining cost per ton.

The rate of output ordinarily will have a greater effect on the indirect or overhead costs per ton than on the direct costs. The indirect charges usually are relatively inflexible and their amount varies little, if any, with variations in production rate. Thus, doubling production rate approximately halves the indirect costs per ton.

Figures 120, *A*, *B*, and *C*, illustrate the effect of production rate upon direct mining cost per ton for open stoping, shrinkage stoping, and cut-and-fill stoping. It is noted that there are two curves for each method, indicating in general two cost ranges resulting from marked differences in mining conditions, wage rates, amount of development work required, or variations in stoping methods. In figure 120, *A*, the difference is due principally to the fact that at

Argonaut, Calif., 1929	Square set	7,800, 1 month	4.50	4.00	2.882	.013	[25] .296	.159	.338	.303	3.991
Bunker Hill & Sullivan, Idaho, 1928.	do	452,345, year	?	?	1.934	.229	(5)	.167	.321	1.246	3.897
Bunker Hill & Sullivan, Idaho, 1931.	do	460,366	?	?	1.875	.204	.113	.165	.331	.766	3.454
Page, Idaho, 1928	do	90,460, year	?	?	2.690	.280	.120	.240	.380	.110	3.820
Park-Utah, Utah	Square set, cut-and-fill	192,250, year	5.25	4.75	3.062	.139	.191	.227	.493	.550	4.662
Tintic Standard, Utah, 1929	Square set	8,758, 1 month	4.75	4.25	6.356	.770	.261	.224	.913	.960	9.484
Gold Hill, Idaho, 1936	do	3,579, 1 month	4.50	4.00	2.820	.410	.200	.110	.320	.590	4.450
Eureka-Standard, Utah, 1933	do	3,737, 1 month	3.50	3.00	4.390	(5)	1.530	.350	.510	.620	7.400
Victoria, British Columbia, 1934.	do	133,195, year	Bonus system		1.004	.063	.108	1.48	.170	.275	1.768
Ray, Ariz., 1928	Block caving	3,243,159, year	Contract and bonus		.3714	.0214	.0351	.0507	.0563	.1417	.6766
Inspiration, Ariz., 1928	do	4,879,646, year	Contract and bonus		.414	.024	.014	.036	.084	.011	.5830
Braden, Chile, 1928	do	5,174,017, year	Mostly contract		.1585	.0128	.0014	.0216	.0476	[26] .1641	.4060
Eureka-Asteroid, Mich., 1929	Sublevel caving	401,680, year [2]	Mostly contract		.927	.134	.143	.128	.068	.137	1.537
Montreal, Wis., 1928	do	450,000, year [2]	Mostly contract		.808	.091	.150	.093	.060	.077	1.279 [27] .060 1.339
Mesabi No. 1, Minn., 1928	Top slicing	209,300, year [2]	Mostly contract		.710	.026	.092	.060	.144	.133	1.165
Marquette No. 5, Mich., 1929	do	555,919, year [2]	Mostly contract		.633	.100	.057	.062	.082	.161	1.095
Marquette No. 2, Mich., 1928	do	240,000 [2]	Mostly contract		.781	.092	.141	.093	.105	.154	1.366
Judge, Utah, 1937	do	6,540 per month	$5.75	$5.25	5.139	.342	.288	.370	1.172	.519	7.830
Miami, Ariz., 1916	do				.528	.045	.064	.065	.130	.048	.880

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[5] Included in other items.
[16] Tons of sorted ore.
[17] Per ton of sorted ore.
[18] Total tons broken.
[19] Per ton broken.
[20] Tons hoisted.
[21] Tons broken.
[22] Includes air compression.
[23] Development in waste not distributed.
[24] Includes liability insurance, $0.21, and overhead (supervision, assaying, engineering office, Social Security taxes), $0.62.
[25] Includes power for air compression.
[26] Includes $0.1398 general expense but not $0.0777 head office expenses.
[27] Deferred rock development.

mines falling along the low-cost curve very little if any development work is required. In figure 120, *C*, the high-cost curve is due to three different conditions—the highest costs are at mines employing resuing, careful sorting in the stopes, and rejection of a large proportion of the rock broken, followed by mines with a higher production but in heavy ground requiring use of the stringer set-and-fill method or considerable square-set timbering in conjunction with cut-and-fill stoping.

Using averages computed from data in Bureau of Mines information circulars, Mears [55] has derived the following figures (table 42) showing the effect of production rate upon direct mining costs.

TABLE 42.—*Average direct cost of mining per ton of ore* (*Mears*)

Daily production, tons	Labor		Supervision	Compressed air, drills, and steel, cost	Power		Explosives		Timber		Supplies, cost	Total cost	Percentage of cost
	Cost	Man-hours			Cost	Kw.-hr.	Cost	Pounds	Cost	Feet			
25													
Development	$1.08	1.8	$0.06	$0.34	------	----	$0.25	1.4	$0.17	6.0	$0.15	$2.05	28.5
Mining	2.78	5.3	.26	.41	$0.69	28	.38	2.1	.35	12.4	.23	5.10	71.5
Total direct	3.86	-----	.32	.75	.69	----	.63	-----	.52	------	.38	7.15	------
Percent	54.0	-----	4.5	10.5	9.6	----	8.8	-----	7.3	------	5.3	------	------
50													
Development	$0.81	1.4	$0.05	$0.22	------	----	$0.20	1.1	$0.12	4.3	$0.12	$1.52	26.5
Mining	2.37	4.6	.23	.32	$0.48	23	.31	1.7	.28	10.4	.23	4.22	73.5
Total direct	3.18	-----	.28	.54	.48	----	.51	-----	.40	------	.35	5.74	------
Percent	55.3	-----	4.9	9.4	8.4	----	8.9	-----	7.0	------	6.1	------	------
100													
Development	$0.66	1.1	$0.04	$0.15	------	----	$0.16	.9	$0.08	3.0	$0.10	$1.19	24.6
Mining	2.12	3.2	.20	.26	$0.35	19	.25	1.4	.24	9.2	.22	3.64	75.4
Total direct	2.78	-----	.24	.41	.35	----	.41	-----	.32	------	.32	4.83	------
Percent	57.5	-----	5.0	8.5	7.3	----	8.5	-----	6.6	------	------	------	------
200													
Development	$0.52	.85	$0.04	$0.10	------	----	$0.13	.7	$0.05	2.0	$0.08	$0.92	22.6
Mining	1.90	3.05	.17	.21	$0.24	16	.20	1.2	.21	8.0	.22	3.15	77.4
Total direct	2.42	-----	.21	.31	.24	----	.33	-----	.26	------	.30	4.07	------
Percent	59.5	-----	5.1	7.6	5.9	----	8.1	-----	6.4	------	.74	------	------
500													
Development	$0.38	.6	$0.03	$0.06	------	----	$0.09	.5	$0.03	1.3	$0.06	$0.65	20.0
Mining	1.62	3.0	.15	.14	$0.15	13	.15	1.0	.17	6.4	.22	2.60	80.0
Total direct	2.00	-----	.18	.20	.15	----	.24	-----	.20	------	.28	3.25	------
Percent	61.5	-----	5.5	6.2	4.6	----	7.4	-----	6.2	------	8.6	------	------
1,000													
Development	$0.30	.5	$0.03	$0.04	------	----	$0.07	.5	$0.02	1.0	$0.05	$0.51	18.6
Mining	1.38	2.7	.13	.11	$0.12	12	.13	.8	.14	6.5	.22	2.23	81.4
Total direct	1.68	-----	.16	.15	.12	----	.20	-----	.16	------	.27	2.74	------
Percent	61.3	-----	5.8	5.5	4.4	----	7.3	-----	5.8	------	9.9	------	------

These figures do not apply to any specific mine or mining methods and make no allowances for width and characteristics of the deposits. Enough mines were included so that the averages should indicate roughly the percentage of variation in costs with different rates of output.

For the purpose of comparing results at different mines where wage rates and supply costs differ considerably, costs expressed in terms of man-hours of labor per ton and tons per man-shift and in units of explosives, timber and power consumption are often more significant than dollar costs. Table 43 gives costs on this basis.

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[55] Mears, Henry S., Average Cost Figures of Mining and Milling Ores: Min. Jour. (Phoenix, Ariz.), Sept. 15, 1936, p. 8.

TABLE 43.—*Total direct mining costs in units of labor, supplies, and power*

Mine and year	Principal stoping method	Rate of output per month or year, tons	Direct unit mining costs per ton of ore mined								All supplies and power, percent of total dollar cost
			Labor, man-hours					Explosives, pounds	Power, kw.-hr.	Timber, board feet	
			Development	Stoping only	Other underground	Surface	Total				
Mineville, N. Y., 1927	Open, with pillars	692,726, year [1]	0.080	0.716	0.277		1.073	0.937	10.91	None	39.60
Tri-State No. 1, 1928	do	11,460, month	0.784		.557		1.341	.750	7.43	None	36.65
Tri-State No. 2, 1928	do	9,730, month	.642		0.234	--------	[2] .876	1.265	5.75	None	42.00
Tri-State No. 3, 1927	do	132,384, year	.401		.435	0.069	.905	.875	4.17	None	35.31
Waco, Mo., 1927–28	do	182,140 per year	.634		.407	--------	[2] 1.041	.594	4.99	None	39.94
Barr, Kans., 1929	do	11,355, month	.502		.386	--------	[2] .888	.805	6.09	None	44.5
Hartley-Grantham, Kans., 1929	do	10,000 per month	.490		.255	.025	.770	.741	1.89	None	36.2
Hartley, Kans., 1932	do	30,000, month	.460		.213	.040	.713	1.120	4.25	None	39.0
No. 8, Southeast Missouri, 1928	do	168,089, year	.490		.385	--------	[2] .875	.634	4.56	None	--------
Mine X, Southeast Missouri, 1929	do	54,000 per month	.135	0.429	.263	--------	[2] .827	.550	10.30	None	28.0
Marquette No. 1, Mich., 1928	do	420,000, year [1]	.194	.703	.416	--------	[2] 1.313	.901	11.56	0.445	35.0
Burra-Burra, Tenn., 1928	Sublevel stoping	473,292, year	.242	.606	.394	.074	1.316	.755	7.92	.610	34.4
Menominee No. 1, Mich., 1928	do	200,000, year [1]	.099	.402	.284	--------	[2] .785	.825	11.80	.0306	38.0
Spring Hill, Mont., 1929–30	Modified sublevel stoping	4,814 per month	.250	1.081	.382	.191	1.904	1.262	10.43	.385	27.3
Mt. Isa, Queensland, 1935–36	Sublevel stoping	703,924, 1 year	.225	.713	.774	.355	2.067	.960	19.52	1.43	41.2
Mascot, Tenn., 1929	Open, with pillars	52,860 per month	0.316		.515	--------	[2] .831	.502	9.68	--------	--------
El Potosi, Mexico, 1933	Open, underhand benches	53,500, 1 month	1.709			.937	2.646	.603	8.23	--------	57.95
Mary, Tenn., 1928	do	108,519, year	.305	1.023	0.904	.633	2.865	.782	10.50	--------	40.80
Isabella, Tenn, 1928	do	74,000, year	.225	.638	.381	--------	[2] 1.244	.654	6.42	--------	31.10
Vanadium, Colo., 1931	Open, with pillars	4,875, month	.130	1.440	.300	--------	[2] 1.870	1.15	.75	--------	40.00
Presidio, Tex., 1929	do	54,524, year	.930	2.740	2.100	.448	6.218	2.00	23.70	--------	26.68
Granada, Quebec, 1932	Open, overhand	3,104, 1 month	1.210	1.675	1.651	--------	[2] 4.536	5.887	35.4	2.48	43.7
Teziutlan, Mexico, 1931	Open and cut-and-fill	65,764, year	1.184	4.298	5.472	1.027	11.981	1.469	62.14	7.997	30.6
Edwards, N. Y., 1930	Open and some shrinkage	72,742, year	.045	.832	.626	(4)	1.503	?	61.09	--------	47.0
Pilgrim, Ariz., 1937	Open with pillars	2,593, 1 month	1.556	1.418	(3)	.540	3.514	--------	--------	--------	--------
Harmony, Idaho, 1929	Shrinkage	3,800, month	1.25		.61	--------	[2] 1.86	2.05	10.10	--------	46.80
Nevada - Massachusetts, Nev., 1928.	do	40,924, year [5]	.501	1.459	1.091	.142	3.193	1.79	10.48	[6] 3.63 [7] .61	43.98
Hillside, Ill., 1929	Shrinkage with some square set	51,208, year	1.504		1.226	--------	[2] 2.730	1.800	10.4	[7] 1.50	30.00
Daisy, Ill., 1929	Shrinkage and stulled stopes	42,000, year	.820	1.200	1.900	.178	4.178	2.50	24.7	[6] 6.00	32.00
Cortez, Nev., 1929	Shrinkage	43,806, year	1.136	1.585	.641	--------	[2] 3.362	3.30	31.77	[6] 2.27	28.02
Teck-Hughes, Ontario, 1928	do	36,629, 1 month	.179	1.517	.801	--------	[2] 2.497	2.36	19.56	[6] 3.52	34.40
Kirkland Lake, Ontario, 1930	do	4,108, 1 month	2.700	1.450	2.680	--------	[2] 6.830	4.72	84.60	[6] 4.23	38.00
Elkoro, Nev., 1930	Shrinkage and cut-and-fill	57,539, year	1.017	1.373	.909	.117	3.416	2.23	10.92	[6] 1.687 [7] .990	--------

Vipond, Ontario, 1929-30	Shrinkage	104,381, year	.424	1.176	1.226		[2] 2.826	1.78	28.00	[6].42 [7].70	43.34
Porcupine-United, Ontario, 1929-30	do	600 per month	.174	3.130	1.406	1.600	6.310	.926	91.60	[6] 2.67	24.90
Engels, Calif., 1928	do	303,301, year	.409	.598	.528	(3)	1.535	1.379	9.64	[6] 1.231 [7].074	34.48
Marquette No. 4, Mich., 1929	Semishrinkage	117,224, year [1]	.206	.599	.380	.403	1.588	1.012	13.83	[6].007	40.20
Mt. Hope, N. J., 1930	Shrinkage	176,474, year [1]	.328	.115	.686	.076	1.205	1.147	13.33		19.60
Black Butte, Oreg., 1929	do	3,626, month	0.862		.610	.241	1.713	1.160	18.45		
Eustis, Quebec, 1934	do	71,709, year	2.725			.423	3.148	4.427	19.31	[6] 2.975	34.41
Alaska-Juneau, Alaska, 1928	Forced caving	3,670,910, year	.048	.086	0.132	.106	.372	.40	1.61		19.17
Brittania East Bluff, British Columbia.	do	398,664, year	.034	.165	.068		[2].267	.76	6.59	[6].65	44.40
Cold Springs, Colo., 1931	Resuing (cut-and-fill)	141 per month [8]	[9] 2.760	[9] 5.690	[9] 9.540		[9] 17.990	[9] 4.48	[9] 61.60	[6] 4.4 [7] 42.3	37.00
Do	do	911.4 per month [10]	[11].424	[11].874	[11] 1.464		[11] 2.762	[11].688	[11] 9.46	[6].68 [7] 6.50	
Block P, Mont., 1929	Cut-and-fill, resuing in thin ore	106,242, year [8]	[9].284	[9] 3.956	[9] 1.110	[9].469	[9] 5.819	[9] 7.878	[9] 17.46	[6] 3.96 [7] 7.54	27.37
Matahambre, Cuba, 1928	Cut-and-fill, some square set	364,746, year	1.000	2.280	1.030	.037	4.347	1.020	14.01	[6] 1.15	40.22
Teziutlan, Mexico, 1930-31	Cut-and-fill, some open stopes	65,764, year	1.184	4.298	5.472	1.027	11.981	1.469	62.14	[6] 7.997	30.60
Pecos, N. Mex., 1929	Cut-and-fill, shrinkage, square-set.	217,121 [12]	1.174	3.071	1.595		[2] 5.840				
Champion, Mich., 1930	Sublevel-inclined, cut-and-fill	38,700 per month	.450	1.490	1.100	.123	3.163	1.300	21.70	[6] 4.30	34.00
Morning, Idaho, 1928	Stringer-set-and-fill	276,890, year	2.837		1.325		[2] 4.162	.964	39.37	[6] 12.75	27.50
Silver King Coalition, Utah, 1929	Stringer-set-and-fill and square set.	180,208, year	1.689	3.771	2.595		8.055	3.607	77.70		
Colorada, Mexico, 1929	Cut-and-fill	43,713 average	2.665		.954	.740	4.359	1.370	11.29	[6] 7.19	42.16
Park City Consolidated, Utah, 1935.	Cut-and-fill and stulled stopes	6,219 per month	1.073	1.437	.698		[2] 3.208	.949	19.60	[6] 10.26	20.40
McIntyre, Ontario, 1931	Cut-and-fill and shrinkage	66,902, 1 month [12]	.112	1.520	.718	.267	2.617	1.090	25.89		37.90
Golden Anchor, Idaho	Resuing	9,547, year	1.780	3.240	1.73		6.750	2.830	47.25	[6] 4.57	26.80
Magma, Ariz., 1928	Cut-and-fill and square set	263,094, year	.502	1.706	2.052	.519	4.779	.90	34.61	[6] 19.00	31.63
Argonaut, Calif., 1929	Square set	7,800, 1 month	.296	2.876	.996		[2] 4.168	1.01	33.08	[6] 7.05 [7] 1.08	27.80
United Verde Extension, Ariz., 1928.	do	275,212, year	(4)	1.653	1.586	.761	4.000			[6] 18.53	
Tintic Standard, Utah, 1929	do	8,758, 1 month	.939	3.241	3.045	1.142	8.367	1.068	46.60	[6] 25.51	31.60
Black Rock, Mont., 1929	do	15,929, 1 month	2.850		1.888	1.481	6.219	1.730	53.70	[6] 12.00 [7] 2.80	30.20
Page, Idaho, 1928	do	90,460, year	2.222		.536		[2] 2.758	1.405	15.07	[6] 13.09	32.50
Ground Hog, N. Mex., 1930	Square set; some cut: and-fill	3,715 per month	1.328	2.140	1.290	1.947	6.705	.900	8.75	[6] 8.12	30.00
Central Eureka, Calif., 1930	Square set	4,143, 1 month	2.746		1.373	.419	4.538	1.010	86.00	[6] 25.78	30.70

[1] Long tons of 2,240 pounds.
[2] Underground only.
[3] Included under other headings.
[4] Included in other underground.
[5] About 15,000 tons more ore broken but not hoisted.
[6] Board feet.
[7] Linear feet.
[8] Tons of sorted ore.
[9] Per ton of sorted ore.
[10] Total tons broken.
[11] Per ton of total ore broken.
[12] Total ore from stoping and development.

TABLE 43.—*Total direct mining costs in units of labor, supplies, and power*—Continued

Mine and year	Principal stoping method	Rate of output per month or year, tons	Direct unit mining costs per ton of ore mined								All supplies and power, percent of total dollar cost
			Labor, man-hours					Explosives, pounds	Power, kw.-hr.	Timber, board feet	
			Development	Stoping only	Other underground	Surface	Total				
Gold Hill, Idaho, 1936	Square set	3,579, 1 month	[13] 0.260	[14] 1.630	1.510	----	[2] 3.400	.920	37.50	[6] 9.27 [7] 2.88	37.00
Humboldt, Ariz., 1928	Block caving	140,000 per month	.548	.127	0.091		.766	.190	----	[6] .25	----
Ray, Ariz., 1928	do	3,243,159, year	([4])	.069	[15] .592	0.077	.738	.319	----	[6] 1.88	----
Braden, Chile, 1928	do	5,174,017, year	0.656			.263	.919	.116	2.91	[6] 1.08	----
Eureka-Asteroid, Mich., 1929	Sublevel caving	401,680, year [1]	[16] .441	[16] .550	[17] .406	.409	1.806	.710	15.62	[6] 3.35	34.00
Montreal, Wis., 1928	do	450,000 year [1]	.157	.397	.674	.108	1.336	.590	[18] 6.24	[6] 2.76	----
No. 20, 1929	do	380,318 year [1]	.200	.365	.406	----	[2] .971	.722	9.38	?	----
No. 16, 1929	Sublevel caving and top slicing	347, 615 [1]	.127	.636	.430	----	[2] 1.193	.421	----	[6] 1.55 [19] .0013	----
Mesabi No. 1, Minn., 1928	Top slicing	209,300, year [1]	0.725		.394	.055	1.174	.395	4.47	[6] 2.83	38.5
Mesabi No. 2, Minn., 1929	do	406,200, year [1]	.200	.515	.439	.383	1.537	.801	2.97	----	32.7
Mesabi No. 3, Minn., 1929	do	337,735, year [1]	None	.585	.146	.167	.898	.694	1.86	[7] 1.16 [19] .0012	39.7
Mesabi No. 4, Minn., 1929	do	315,083, year [1]	None	.540	.237	.212	.989	.463	3.49	[6] 1.06 [7] 1.256 [19] .0018	35.1
Mesabi No. 5, Minn., 1929	do	270,221, year [1]	.007	.617	.369	----	[2] .993	.533	3.36	[6] 1.34 [19] .0016	932.
Mesabi No. 6, Minn., 1929	do	777,109, year [1]	.076	.645	.406	.220	1.347	.562	4.16	[7] 2.21 [19] .00325	29.6
Marquette No. 9, Mich., 1929	do	555,919, year [1]	.091	.533	.279	.112	1.015	.408	12.49	[7] .672	42.1
Marquette No. 11, Mich., 1929	do	275,000, year [1]	.090	.919	.377	.385	1.771	.547	9.39	[6] 1.702 [19] .00394	34.0
Marquette No. 12, Mich., 1929	do	479,812, year [1]	.145	.556	.347	----	[2] 1.048	.437	18.41	[6] 1.117 [7] 1.849 [19] .0047	42.3
Marquette No. 13, Mich., 1929	do	343,147 year [1]	.027	.695	.296	.297	1.315	.365	16.41	[6] 1.376 [7] 3.052	44.7
Judge, Utah, 1937	do	6,540 per month	.690	2.750	2.000	.842	6.282	3.360	14.50	[6] 56.52	34.30

[1] Long tons of 2,240 pounds.
[2] Underground only.
[4] Included with other underground.
[6] Board feet.
[7] Linear feet.
[13] Exploration only.
[14] Development and stoping.
[15] Includes development.
[16] Includes apportioned part of supervision and general labor.
[17] Haulage and hoisting.
[18] Plus 0.32 pounds of coal.
[19] Cord of lagging.

SURFACE MINING

The term "surface mining" is used in this bulletin to cover the mining in surface excavations. It includes placer mining, mining in open glory-hole or "milling" pits, mining and removing ore from open-cuts by hand or with mechanical excavating and transportation equipment, and the removal of capping or overburden to uncover the ores.

In this section of the bulletin are assembled operating data from a number of earlier publications of the Bureau of Mines and some data from other sources. Space does not permit comprehensive discussion of methods, practices, and equipment, and these subjects are therefore considered but briefly.

PLACER MINING

The subject of placer mining is one upon which much has been written in the technical press, handbooks, textbooks, publications of technical societies, and in earlier publications of the Bureau of Mines.[56]

As in the case of lode deposits, methods of working depend largely on natural conditions—the mode of occurrence or type of deposit, the size and grade (value per cubic yard), bedrock gradient, the depth of overburden and pay gravel, the nature of the gravel, particularly whether cemented or not, the quantity of large boulders, the quantity and head of water available, and climatic conditions.

TYPES OF PLACER DEPOSITS

There are two general types of placer deposits—residual placers and transported or alluvial placers. In both, the original source of the gold was in lodes, veinlets, stringers, or stockwerks in solid rock. In the residual deposits, which are relatively unimportant from the standpoint of gold production, the gold occurs in disintegrated rock at or near the original source, and concentration of the gold has been effected by removal in solution of soluble rock constituents and by mechanical washing out of lighter minerals.

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[56] Janin, Charles, Gold Dredging in the United States: Bureau of Mines Bull. 127, 1918, 226 pp.

————, Recent Progress in Thawing of Frozen Gravel in Placer Mining: Bureau of Mines Tech. Paper 309, 1922, 34 pp.

————, Placer-Mining Methods: Bureau of Mines Rept. of Investigations 2315, 1922, 4 pp.

Jennings, Hennen, The History and Development of Gold Dredging in Montana: Bureau of Mines Bull. 121, 1916, 63 pp.

Wimmler, N. L., Placer-Mining Methods and Costs in Alaska: Bureau of Mines Bull. 259, 1927, 236 pp.

Jackson, Chas. F., and Knaebel, John B., Small-Scale Placer-Mining Methods: Bureau of Mines Inf. Circ. 6611, 1932, 17 pp.

Johnson, Fred W., and Jackson, Chas. F., Federal Placer-Mining Laws and Regulations and Small-Scale Placer-Mining Methods: Bureau of Mines Inf. Circ. 6611R, 1938, 37 pp.

Patmon, Charles G., Methods and Costs of Dredging Auriferous Gravels at Lancha Plana, Amador County, Calif.: Bureau of Mines Inf. Circ. 6659, 1932, 16 pp.

Requa, Lawrence K., Description of the Property and Operations at the Lewiston Dredge, Lewiston, Calif.: Bureau of Mines Inf. Circ. 6660, 1932, 14 pp.

Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. Part I. General Information, Hand Shoveling, and Ground Sluicing; Bureau of Mines Inf. Circ. 6786, 1934, 73 pp.

————, Placer Mining in the Western United States. Part II. Hydraulicking, Treatment of Placer Concentrates, and Marketing of Gold: Bureau of Mines Inf. Circ. 6787, 1934, 89 pp.

————, Placer Mining in the Western United States. Part III. Dredging and Other Forms of Mechanical Handling of Gravel, and Drift Mining: Bureau of Mines Inf. Circ. 6788, 1935, 81 pp.

Ross, Charles L., and Gardner, E. D., Placer-Mining Methods of E. T. Fisher Co., Atlantic City, Wyo.: Bureau of Mines Inf. Circ. 6846, 1935, 10 pp.

Gardner, E. D., and Guiteras, Jos. R., Placer Operations of Humphreys Gold Corporation, Clear Creek, Colo.: Bureau of Mines, Inf. Circ. 6961, 1937, 16 pp.

Gardner, E. D., and Allsman, Paul T., Power-Shovel and Dragline Placer Mining: Bureau of Mines Inf. Circ. 7013, 1938, 68 pp.

Lorain, S. H., and Metzger, O. H., Reconnaissance of Placer-Mining Districts in Idaho County, Idaho: Bureau of Mines Inf. Circ. 7023, 1938, 94 pp.

Metzger, O. H., Reconnaissance of Placer Mining in Boise County, Idaho: Bureau of Mines Inf. Circ. 7028, 1938, 34 pp.

Vanderburg, William O., Placer Mining in Nevada: University of Nevada Bull. 4, vol. 30, May 15, 1936, 180 pp.

Transported placers result from the removal of gold-bearing rock debris from its original situation by natural agencies (principally running water), the further disintegration of the rock during transportation with attendant liberation of the gold particles, and redeposition at some distance from the source. By reason of the sorting action of running water, gold, which is of much higher specific gravity than the rock-forming minerals, is separated from them and deposited, under favorable conditions, where the velocity of the water is reduced to a rate such that the gold is dropped but the lighter minerals are carried farther.

Brooks [57] has classified placers largely on the basis of the present position of the deposits.

Classification of placer deposits (Brooks)

Creek placers.—Gravel deposits in the beds and intermediate flood plains of small streams (fig. 1).

Bench placers.—Gravel deposits in ancient stream channels and flood plains that stand 50 to several hundred feet above the present streams. They represent the remnants of the stream beds during earlier stages of development.

Hillside placers.—Gravel deposits intermediate between the creek and bench gravels; their bedrock is slightly above the creek bed and the surface topography shows no indication of benching.

River-bar placers.—Placers on gravel flats in or adjacent to the beds of large streams.

Gravel-plain placers.—Placers found in the gravels of the coastal or other lowland plains

Sea-beach placers.—Placers reconcentrated from the coastal-plain gravels by the waves along the seashore.

Ancient beach placers.—Deposits found on the coastal plain along a line of elevated benches.

Lake-bed placers.—Placers accumulated in the beds of present or ancient lakes that were generally formed by landslides or glacial damming.

Another type of placer is the so-called dry placer occurring in arid regions in poorly assorted angular or subangular gravels over large areas of gulches, hillsides, mesas, and ridges instead of in well-defined channels.

Creek, bench, and river-bar placers have been the most productive. As gold is virtually the only metal recovered in large amounts by placer mining in this country, the following discussions and data apply to gold mining.

METHODS OF WORKING

Placer-mining methods may be classified either according to the methods employed for excavating and transporting the gravel or the method of recovering the gold. Using the methods of excavation and transportation as a basis and employing terms familiar to placer miners, the following classification may be made:

1. Hand shoveling. Gold recovered by (*a*) panning, (*b*) rocking, (*c*) use of the long tom, (*d*) shoveling into boxes (sluicing), or (*e*) dry washing.
2. Ground sluicing (including "booming").
3. Hydraulicking.
4. Excavation by teams or power-operated equipment other than dredges.
5. Dredging.
6. Drift mining (an underground method).

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[57] Brooks, Alfred H., The Mineral Deposits of Alaska: Mineral Resources of Alaska, 1913; Geol. Survey Bull. 592, 1914, pp. 25–32.

HAND METHODS

Hand methods are adapted to the mining of comparatively thin deposits under shallow cover that are too small to justify the capital expenditure required for working by other methods. Excavation is by pick and shovel and the gravel may be washed by panning, rocking, or long tom in the "diggings" or transported by wheelbarrow, pack animals, carts, or trucks to the water and washed by panning, rocker, long tom, or sluice. Since ordinary cheap tools are employed and the washing equipment can be built on the ground at small expense, hand methods are available to those of small means. In this country, however, very few deposits rich enough to pay wages when worked by hand methods remain unworked. A skillful operator can pan carefully about 100 pans, or a little over half a cubic yard of uncemented gravel (measured in place), in 10 hours. Thus, to earn $5 per day, gravel would have to run about $10 per cubic yard.

Rocking requires but little more equipment than panning, and a rocker can be built by anyone having average skill with carpenter's tools. Figure 121, *A*, shows a knock-down rocker, which differs from other forms only in details of construction and which can be varied to suit conditions. The rocker may be operated by one man but preferably by two, one of whom operates the rocker while the other digs the gravel and brings it to the rocker. Where water is scarce, the rocker often is placed alongside a small sump or pit, from which water is taken with a dipper and poured over the gravel in the sieve box. If water can be piped to the box, the labor of operating the rocker is reduced considerably and more gravel can be washed per shift. According to Purington,[58] two men rocking steadily can wash 3 to 5 cubic yards per 10-hour day, the exact amount depending upon the nature of the gravel, the distance it has to be carried, and the water supply. The gold is caught in the canvas apron under the sieve box and behind the riffles in the box, whence it is removed periodically or whenever the concentrates build up to the level of the tops of the riffles.

In recent years some miners have added mechanical drives to the rockers, increasing their capacity and lessening the manual labor required. In one instance [59] an eccentric arm attached to the rocker was driven by a small gasoline engine, and two men were able to handle an average of 1 cubic yard per hour; where the gravel was free from clay, the capacity was said to be as high as 3 cubic yards per hour. The cost of the rocker and drive was $160. A number of patented machines constructed on similar principles are on the market, cost $225 to $700, and have advertised capacities of ½ to 2½ cubic yards per hour.

The long tom (figure 121, *B*) usually has a somewhat greater capacity than the rocker and does not require hand labor or power for rocking but requires a good supply of running water and often does not give as complete recovery of the gold. Wilson [60] states that two men, one shoveling into the tom and one forking out the large stones and keeping it clear, can wash 6 cubic yards of ordinary

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[58] Purington, C. W., Methods and Costs of Gravel and Placer Mining in Alaska: Geol. Survey Bull. 263, 1905, 273 pp.

[59] Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. Part I. General Information, Hand Shoveling, and Ground Sluicing: Bureau of Mines Inf. Circ. 6786, 1934, p. 50.

[60] Wilson, E. B., Hydraulic and Placer Mining: John Wiley & Sons, Inc., New York, 3d ed., 1918, 425 pp.

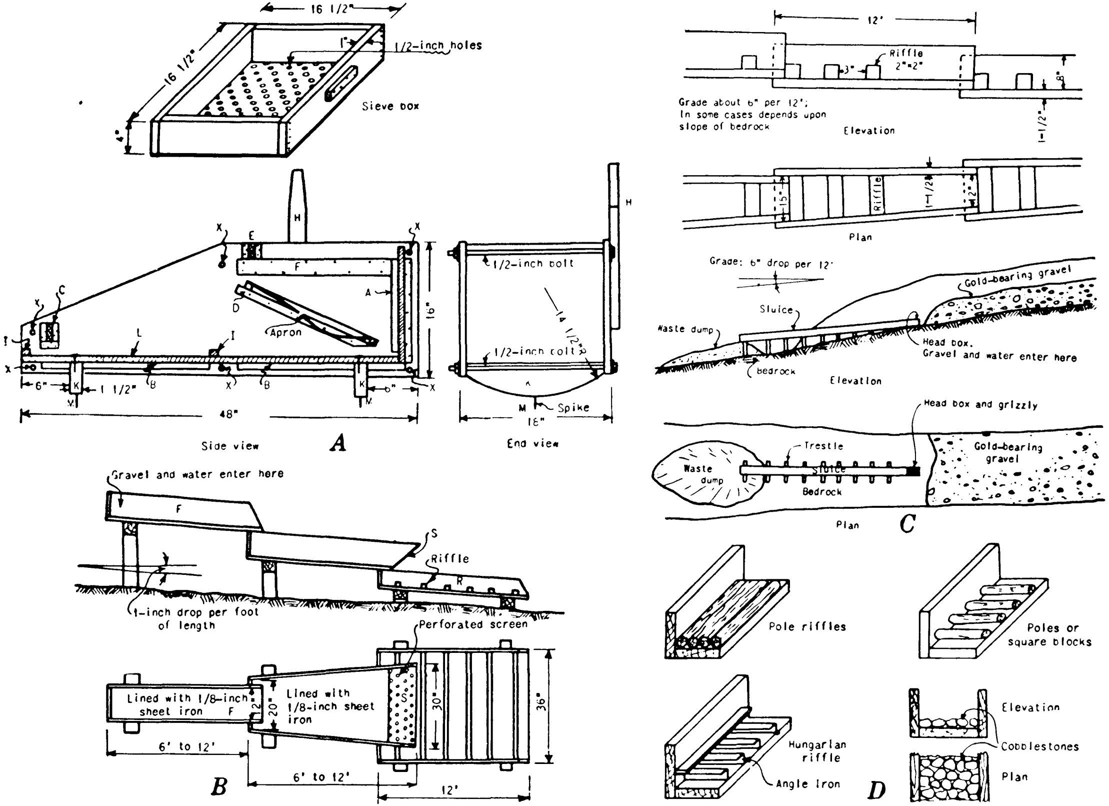
16 1/2"
16 1/2"
1"
1/2-inch holes
4"
Sieve box
H
E
X
F
A
D
Apron
C
I
L
B
K
M
6"
1 1/2"
6"
16"
48"
Side view
A
1/2-inch bolt
14 1/2"R
Spike
18"
End view
Gravel and water enter here
F
S
Riffle
R
1-inch drop per foot of length
Perforated screen
Lined with 1/8-inch sheet iron F
2"
20"
Lined with 1/8-inch sheet iron
S
30"
36"
6' to 12'
6' to 12'
12'
B
12'
Riffle 2"×2"
3"
8"
Grade about 6" per 12'; In some cases depends upon slope of bedrock
Elevation
1-1/2"
15"
Riffle
1-1/2"
12"
Plan
Grade: 6" drop per 12'
Gold-bearing gravel
Sluice
Waste dump
Head box. Gravel and water enter here
bedrock
Elevation
Head box and grizzly
Trestle
Waste dump
Sluice
Bedrock
Gold-bearing gravel
Plan
C
Pole riffles
Poles or square blocks
Hungarian riffle
Angle iron
Elevation
Cobblestones
Plan
D

FIGURE 121.—*A*, Sketch of knock-down rocker. (From Eng. and Min. Jour.) *B*, Long tom. *C*, Construction of sluice boxes and typical lay-out for hand work. (After W. A. Staley.) *D*, Common types of riffles.

gravel or 3 to 4 yards of cemented gravel in 10 hours. Toms are used rarely now in the United States; where running water and grade are available a simple sluice is as effective and requires less labor.

In operation the gravel is shoveled into the tom (or flume immediately ahead of it), where the fine material is washed through the screen and the larger rocks are forked out and discarded. The gold and heavy sands are caught behind the riffles, and when these become filled up the concentrates are removed and cleaned in a gold pan. Mercury sometimes is added to the box to catch fine gold.

Surf washers are similar to long toms but usually are shorter and wider. They are set so that the incoming surf rushes up the sluices, washes material from the screen box or hopper, and, upon retreating, carries it over the riffles and plates. The duty per man averages about 3 to 5 cubic yards in 10 hours. One man can tend two surf washers, and in one recorded instance 8 cubic yards were handled in 10 hours.

Hand-shoveling into boxes or sluicing requires a plentiful water supply and a good bedrock gradient or slope to carry away the washed gravel. The method is adapted to rich gravel up to 6 or 8 feet in depth or to gravel that has already been partly concentrated by ground sluicing or booming. The gravel is loosened by picking and is shoveled by hand into the head of the sluice, which consists of a series of telescoping riffled boxes (fig. 121, *C*, which also shows a typical lay-out for hand work).

Riffles are constructed in various ways, four common types being shown in figure 121, *D*. The quantity of water available will control the scale of operations and the size of sluice to be used. A minimum flow of 15 to 20 miner's inches (170 to 225 gallons per minute) is required for a 12-inch sluice set on a steep gradient. Boxes are 8 to 24 inches wide, and there may be from three to as many boxes as are required to carry the tailings to a suitable dumping ground. Concentrates from the sluices are panned or washed in a rocker to effect the final separation of the gold. The amount of water required for sluicing varies; about 1.3 to 3 cubic yards of gravel can be handled per 24 hours with a flow of 1 cubic foot of water per minute, equivalent to about 3,600 to 8,300 gallons of water per cubic yard. Young[61] cites two examples of sluice capacities, as follows:

Width of sluice, inches	*Depth of flow, inches*	*Gradient, percent*	*Water flow per minute, cubic feet*	*Gravel washed per 24 hours, cubic yards*
10–12	6–7	4. 16	45	67. 5–135
12–14	10	6. 2	100	150 –300

Under ideal conditions a good miner can shovel 15 cubic yards of gravel per shift into boxes elevated not more than 3 or 4 feet above bedrock. Wimmler[62] states that 7 or 8 cubic yards per man-shift constitutes an average performance in Alaska.

Table 44, reproduced from Information Circular 6786 by E. D. Gardner and C. H. Johnson, gives data on shoveling into boxes, which are typical of this method of working and show wide variations in working costs under different conditions.

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[61] Young, George J., Elements of Mining; McGraw-Hill Book Co., Inc., New York, 1st ed., 1916, p. 351.
[62] Wimmler, Norman L., Work cited in footnote 56 (p. 325).

TABLE 44.—*Representative examples of shoveling into boxes*

	Oroville	Jones	Peshastin	Minnick	Blackhawk	Greetan	DeWitt
Operator of mine		J. B. Jones				O. F. Greetan	H. DeWitt.
Address	Oroville, Calif.	Oroville, Wash	Blewett, Wash	Blewett, Wash	Blackhawk, Colo	Bearmouth, Mont.	Wilcox, Ariz.
Location of mine	Feather River	Mary Ann Creek	Peshastin Creek	Peshastin Creek	North Clear Creek	Bear Creek	Gold Gulch.
Number of men working	2	4	4	2	2	2	1.
Shift hours	9	9	9	8	8	8	8.
Depth of gravel feet	2	6[1]	6	2½	4½	6	
Character of gravel	Loose top dirt	Tight	Tight	Tight	Loose	Medium	Tight.
Percentage of boulders over 6 inches in diameter in gravel.	0	30	20	50	10	5	
Character of bedrock	None	Clay	Even	Very rough	None	Even	Clay.
Width of sluice boxes inches	8	8	12	12	10	8	18.
Grade inches per foot	1	1	7/16	½		6/10	
Length of each box feet	12	12	16	12	12	10	22.
Boxes number	2	8	4	3	3	6	1.
Type of riffles	Hungarian	Steel matting over coco matting.	Hungarian pole	Wire screen over Brussels carpet.	Wire screen over carpet.	Wooden cross	Wooden cross.
Size of riffles inches	1 by 1¼		Hungarian 3/16 by 1¼; pole 3.			1 by 1	1 by 1.
Spacing of riffles do	1		1			4	1.
Total length of riffles feet	12	8	64	12	12	28	18.
Height water pumped do	10	37	0	0	0	0	0.
Percentage of material rehandled	5	10	0	50	0	0	200.
Handled per day cubic yards	20[2] to 30[3]	25	20	2	6½	9½	1.
Handled per man-shift do	10[2] to 15[3]	6¼	5	1	3¼	4¾	1.
Gasoline used per cubic yard gallons	3	6	0	0	0	0	1½.
Estimated cost of labor and gasoline per cubic yard.[4]	$0.36 to $0.26	$0.60	$0.60	$3.00	$0.92	$0.63	$3.30.
Remarks	(5)	Good workmen	Good workmen	(6)		(7)	

[1] Top 2 feet consists of soil.
[2] With 10 percent plus 1½ inches.
[3] With no oversize.
[4] Assuming labor at $3 per shift.
[5] All material over 1¼ inches in size screened out at head box.
[6] Stumps in gravel and rock ledges above bedrock.
[7] Only lower 18 inches of gravel washed.

Dry washing usually has failed or been indifferently successful in this country. Winnowing as practiced in Western Australia, where the sand is very dry and there are strong prevailing winds, has not worked well here because of retained moisture in the sands and because of insufficient wind. For successful dry washing, the gravel must be completely dry and thoroughly disintegrated. Machines similar to that shown in figure 122 consist of screens and flat-sloping, riffled canvas decks, under which air pulsations are produced by bellows or other means to agitate the sand and work it down the deck over the riffles, the gold and other heavy minerals being caught behind the riffles. The machine illustrated was operated to give 250 pulsations of air per minute, and the capacity was reported to be about 0.8 cubic yard per hour, bank measure.

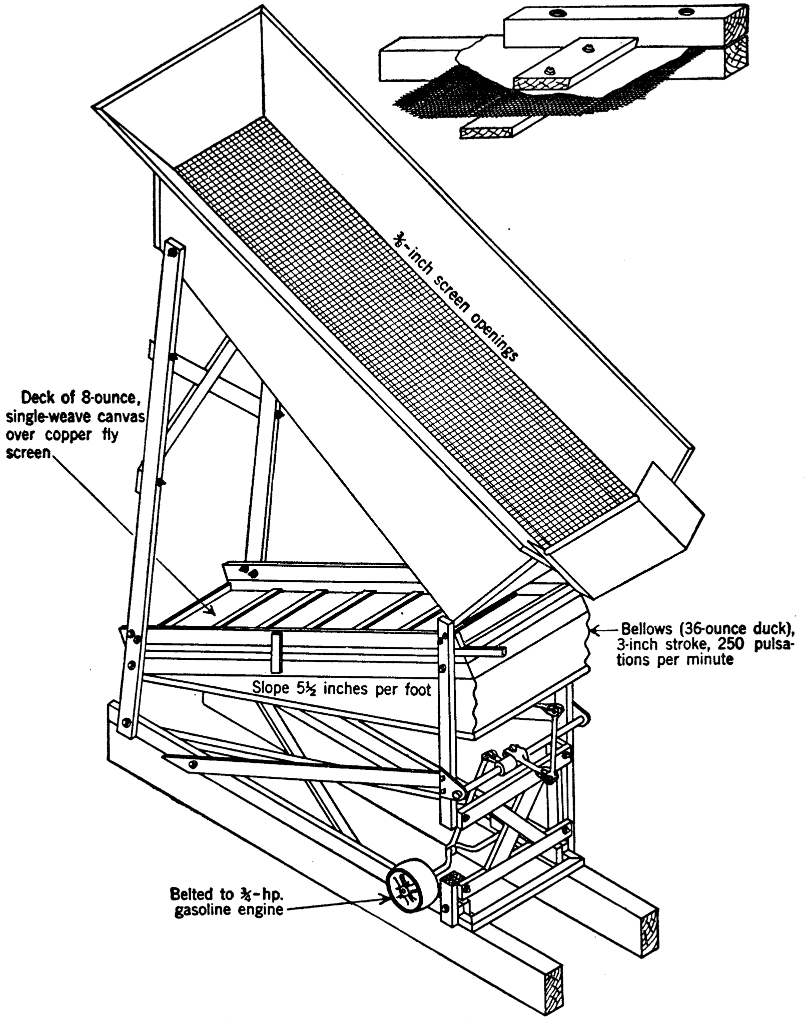
⅜-inch screen openings
Deck of 8-ounce, single-weave canvas over copper fly screen
Bellows (36-ounce duck), 3-inch stroke, 250 pulsations per minute
Slope 5½ inches per foot
Belted to ⅜-hp. gasoline engine

FIGURE 122.—Power-driven dry washer, capacity 0.8 cubic yard per hour, Goler Gulch, Calif. (June 1932).

GROUND SLUICING

In ground sluicing the gravel is excavated by running water not under pressure. Booming is a variation of ground sluicing in which water is stored in reservoirs and is intermittently discharged in large volumes over short periods to obtain the maximum erosional and transporting effect on the gravel. The most favorable conditions for ground sluicing usually occur on the benches and upper reaches of creeks where pipelines, flumes, or ditches, which add greatly to installation costs, are not required. The method is not efficient if the gravel is cemented and, although applicable to gravel containing boulders, considerable additional work is required for handling. Ground sluicing and booming often have been employed to remove barren or very low-grade, light material covering the pay gravels during high-water seasons in conjunction with other methods for mining the gravel.

Excavation by running water may be accomplished in two ways, depending upon surface and bedrock topography and gradients and thickness of the deposit—(1) by diverting the stream to undercut the bank and carry away the caving gravel and (2) by running it over the upper face or crest of the bank, the gravel being broken by the cascading effect. Figure 123 shows four typical ground-sluicing operations. At *A* and *C* are shown adaptations of the first method and at *B* and *D* of the second method. In *C* is shown a pipe line carrying water under a 60-foot head and terminating in a fire hose and nozzle employed to remove the top soil and to assist in undercutting the bank. Booming was employed in the operation shown at *D*, where the gravel bed was 18 to 24 feet deep and 90 feet wide. The bottom 6 feet was fairly tight and by overcasting the water it scoured away this stratum.

The water-borne material is deflected to sluice boxes, in which the gold and heavy minerals collect behind or between the riffles and are later cleaned up in the usual manner by first running clear water through the sluice until it is free from gravel and then removing the riffles section by section, beginning at the upper end, while a light stream of water is passed through. Boulders not transported by the waters must be moved aside to permit cleaning the bedrock beneath them and considerable labor may be required for this purpose. At *D*, figure 123, is shown a derrick for handling large boulders; at other places they are blasted or removed by hand or by cableways or other mechanical means.

In ground sluicing and, indeed, in all placer work the bedrock must be cleaned carefully to recover gold that naturally tends to collect in cracks and other irregularities of its surface.

Table 45 is abstracted from data given in Information Circular 6786 on a number of ground-sluicing and booming operations. Wide variations in costs are shown, and it may be noted that whereas it was stated previously that for shoveling into boxes (water used only for washing the gravel) about 1.3 to 3 cubic yards of gravel could be handled per 24 hours with a flow of 1 cubic foot of water per minute (1.95 to 4.5 cubic yards per miner's inch per day), when water was used for ground sluicing and booming the duty per miner's inch per day at the tabulated mines ranged from 0.04 to 1.7 cubic yards. From these data it appears that ground sluicing and booming usually may require roughly 5 to 10 times as much water as shoveling in.

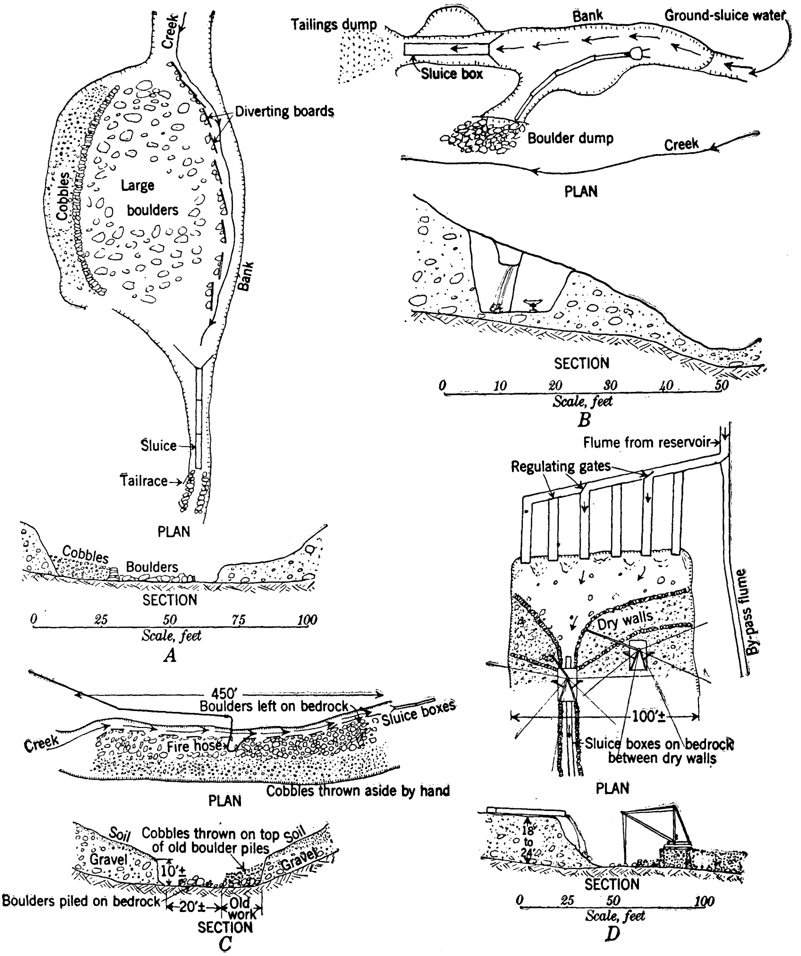
Creek
Diverting boards
Cobbles
Large boulders
Bank
Sluice
Tailrace
PLAN
Cobbles
Boulders
SECTION
0 25 50 75 100
Scale, feet
A
Tailings dump
Bank
Ground-sluice water
Sluice box
Boulder dump
Creek
PLAN
SECTION
0 10 20 30 40 50
Scale, feet
B
450′
Boulders left on bedrock
Sluice boxes
Creek
Fire hose
Cobbles thrown aside by hand
PLAN
Soil
Cobbles thrown on top of old boulder piles
Soil
Gravel
10′±
Gravel
Boulders piled on bedrock
20′±
Old work
SECTION
C
Flume from reservoir
Regulating gates
Dry walls
By-pass flume
100′±
Sluice boxes on bedrock between dry walls
PLAN
18′ to 24′
SECTION
0 25 50 100
Scale, feet
D

FIGURE 123.—Layouts of ground-sluice mines. *A*, Ravano mine, Laurin, Mont. *B*, Rundle mine, Blackhawk, Colo. *C*, Camp Bird mine, Laurin, Mont. *D*, Harvey mine, Lincoln, Mont.

TABLE 45.—*Data on ground sluicing and booming operations, 1932*

Mine and situation	Gravel							Water supply, average miner's inches[1]	Sluices			Riffles		Cost per cubic yard washed		
	Thickness, feet	Boulders over 6 inches, percent	Clay, percent	Volume washed, cu. yds.					Size, inches		Grade, inches per foot	Type	Length, feet	Labor[2]	Supplies	Total
				Season	Day	Per day per miner's inch	Per man-shift		Width	Depth						
Morgan, Colo	5	20	5	165	2.75	0.04	2.75	70	18	12	1.2	Wire screen on carpet	20	$1.27	$0.02	$1.29
Ravano, Mont	6	25	0	775	9.0	.30	9.0	[3] 30	---	---	.5	Pole	36	.39	---	.39
Bar No. 1, Colo	6	10	0	300	9.0	.15	3.0	60	12	8	1.5	Wooden cross on burlap	36	1.17	.02	1.19
Osborne, Mont	15	15	0	1,400	53.0	.66	9.0	80	[4] 13	13	.33	Pole	84	.39	.03	.42
Rundle, Colo	8	15	2	174	3.0	.05	3.0	60	24	12	1.00	Cross on wire over corduroy.	16	1.17	.02	1.19
Bar No. 2, Colo	6	10	0	400	24.0	.40	12.0	60	18	12	.25	Cross on burlap	24	.28	.03	.31
Kamloops, Colo	18	10	0	[5] 4,000	73.0	.73	18.0	100	30	20	.25	Wooden cross	250	.20	.04	[6] .24
Willow Creek, N. Mex	20	10	1	[7] 1,030	16.0	.23	4.0	[8] 70	18	10	.50	Wooden block	600	.87	.04	.91
Camp Bird, Mont	10	15	0	2,400	36.0	.51	18.0	[9] 70	22	16	1.00	Pole	36	.20	.02	.22
Bennet, Mont	15	10	2	1,150	17.0	1.70	17.0	[10] 10	36	18	.75	do	36	.21	.01	.22
Harvey, Mont	22	35	5	3,800	32.0	.127	32.0	[11] 300	38	36	.75	do	96	.11	.03	.14
Magnus, Wash	20	25	1	[12] 6,000	67	.134	7.0	[13] 500	48	36	.60	Steel rails	400	.50	.04	.54

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[1] One second-foot=40 miner's inches, or 1½ cubic feet per minute=1 miner's inch.
[2] Labor figured at $3.50 per day.
[3] Booming, using 150 miner's inches from reservoir for 4 hours.
[4] 24 inches when more water available.
[5] To July 17.
[6] Not including rental of shovel or supervision.
[7] To July 20.
[8] Booming, using 300 miner's inches from reservoir for 27 minutes.
[9] Booming, using 650 miner's inches for 30 minutes.
[10] Booming, using 1,600 miner's inches for 1½ minutes.
[11] Average 700 first 52 days; booming used 7,500 miner's inches from reservoir once every hour, for 2½-minute periods.
[12] To June 23.
[13] Booming used 4,000 miner's inches every 1¾ hours for 15-minute periods.

HYDRAULICKING

In hydraulic mining a stream of water under high head is employed to break down the bank and excavate the gravel and to sweep it to the sluices. Additional running water (termed "bank" or "by-wash" water) usually is required to wash it to and through the sluices. The gold is recovered in part from cleaning the bedrock after the gravel has been removed but principally from clean-up of the riffled sluice boxes.

An adequate supply of water under high pressure is therefore required, and the construction of dams, ditches, flumes, or pipe lines or a combination of these is necessary to convey the water to the mine or pit. Pressure sometimes is supplied by power-driven pumps rather than by natural heads, but this adds substantially to the mining cost. The excavating power of the water varies with the pressure and also the quantity, and yardage that can be handled per day will vary with this and with the total quantity of water (pressure water plus running or "by-wash" water). The cutting and sweeping capacity of the pressure water is usually greater than its carrying capacity after it becomes merely a stream of running water, and therefore additional "bank" water generally is required to wash the gravel through the sluices and ultimately to the tailings pile. If the "by-wash" water is run over the crest of the bank, as in ground sluicing, it may aid materially in cutting down the bank and carrying the gravel to the sluices.

Hydraulic mining is employed principally for excavating thick deposits, but in Alaska it has been used widely for mining creek placers and bench deposits, many of which are but slightly above the creek level. Hydraulicking or "piping" also is used to remove the overburden from the pay gravels, the latter then being mined by other methods. For good results the bedrock slope should be at least 2 percent for finely divided material, 4 or 5 percent for medium-size material, and more for coarse material. If there is not enough natural slope for moving the gravel to and through the sluices and for disposal of the tailings, it becomes necessary to sweep it to a pit or sump and elevate it by means of Ruble or hydraulic elevators or by mechanical means to sluices supported on trestles. Hydraulic mining usually is seasonal, being restricted to periods of maximum run-off.

Pressure water is piped to the pit, usually from a penstock, and is directed against the gravel bank by nozzles termed "giants" or "monitors." These can be swung to direct the jet against desired points in the bank or to sweep the excavated gravel along the bedrock to the sluice head or sump, as the case may be. Table 46 gives data on flow of water through giants.

TABLE 46.—*Flow of water through giants*[1]

Giant No.	Diameter of nozzle, inches	Effective head, feet							
		100		200		300		400	
		Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches	Cubic feet per second	Miner's inches
0	1⅛	0.6	22	0.8	31	----------	----------	----------	----------
0	1⅜	.8	33	1.2	47	----------	----------	----------	----------
1	2	1.6	63	2.2	89	2.7	109	3.1	125
1	3	3.0	120	4.3	173	5.3	213	6.4	257
2	3	3.3	133	4.7	187	5.7	227	6.6	267
2	4	5.6	227	8.3	333	10.3	410	11.9	477
3	3	3.7	148	5.0	200	6.5	245	7.1	283
3	4	6.0	240	8.6	343	10.6	423	12.2	488
4	4	6.3	253	8.9	357	10.9	437	12.6	504
4	6	13.3	535	19.2	770	23.7	950	27.5	1,100
5	5	9.8	395	13.9	560	16.7	670	19.7	790
5	6	13.5	540	19.3	770	23.8	950	27.7	1,110
6	6	13.8	550	19.6	780	24.1	960	27.9	1,120
6	7	18.7	750	26.7	1,070	33.2	1,330	37.7	1,510
7	6	14.2	570	20.0	800	24.5	980	28.3	1,130
7	7	19.0	760	26.9	1,080	33.3	1,330	38.0	1,520
8	7	19.2	770	27.2	1,090	33.8	1,350	38.3	1,530
8	8	15.2	1,010	35.3	1,410	43.7	1,750	48.7	1,950
9	9	32.0	1,280	45.0	1,800	55.3	2,210	63.7	2,550
9	10	39.3	1,570	55.3	2,210	68.2	2,730	78.7	3,140

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[1] Adapted from table in catalog of Joshua Henry Iron Works, San Francisco, Calif.

Table 47 contains operating data on a number of hydraulic mines in the Western United States that are believed to be typical of recent operations. These data are from Information Circular 6787.

TABLE 47.—*Operating data on hydraulic mines*

Mine and situation	Gravel					Water supply, miner's inches				Pressure water head, feet	Duty[1] of total water, cubic yards	Nozzles		Sluices		Operating cost per cubic yard		
	Average thickness, feet	Boulders over 12 inches, percent	Volume washed, cubic yards			By wash water	Pressure water	Total water	Hours used per day			Diameter, inches	Number used at one time	Size, inches[2]	Length, feet	Labor and supervision	Supplies	Total
			Season	Day	Per man-shift													
Senger, Calif	6	20	3, 600	40	20	0	400	400	5	225	0. 48	4¼	1	36 x 24	96	$0. 18	$0. 02	$0. 20
Elephant, Calif	[3] 45	3	8, 000	62	20	40	135	175	10	115	. 85	3	1	18 x 16	32	. 20	. 02	. 22
Horton Gulch, Calif	17	5	8, 000	80	40	30	220	250	9	65	. 85	5	1	24 x 18	36	. 09	. 02	. 11
Banner, Calif	15	10	15, 000	------	84	0	300	300	10	100	------	4	2	26 x 24	36	. 04	. 01	. 05
Indian Creek, Calif	9	8	8, 800	[4] 175	[5] 88	0	500	500	5	275	3. 36	5	1	48 x 36	48	. 046	. 03	. 076
Salmon River, Calif	[6] 17	6	29, 000	223	56	0	1, 100	1, 100	9	225	. 54	6	2	36 x 30	150	. 06	. 01	. 07
Jacobs, Calif	[7] 40	5	12, 000	240	120	0	460	460	3	90	4. 15	7	1	48 x 36	120	. 03	. 02	. 05
Omega Hill, Calif	[8] 45	8	75, 000	1, 700	280	200	1, 300	1, 500	10	210	2. 72	6	2	48 x 36	1, 700	. 02	. 02	[9] . 04
Indian Hill, Calif	35	------	100, 000	------	------	220	430	650	9	130	------	4½ and 6	1	40 x 40	288	------	------	[10] . 07
Depot Hill, Calif	60	5	40, 000	------	------	150	450	600	------	160	1. 0	4½ and 5	1	30 x 24	3, 500	------	------	[10] . 115
North Fork, Calif	15	5	150, 000	[11] 770	140	0	900 400	900 400	18 5	400 200	1. 0	7 and 5	2	48 x 40	168	. 03	. 015	. 045
Salyer, Calif	100	------	718, 900	7, 400	500	0	2, 800	2, 800	24	350	2. 64	7	2	60 x 50	350	------	------	. 026
Norton, Oreg	12	12	12, 000	80	40	390	210	600	9	90	. 55	3 and 4	1	20 x 20	100	. 10	. 02	. 12
Salmon Creek, Oreg	25	8	?	260	33	60	90	150	16	150	2. 55	2¼	1	26 x 20	180	. 15	. 05	. 20
Deep Creek, Mont	15	10	[12] 2, 100	57	11	100	100	200	5	135	1. 37	3	1	20 x 18	300	. 41	. 01	. 42
Yellowstone, Mont	30	5	9, 000	100	31	360	240	600	18	175	. 22	4	1	23 x 24	500	. 11	. 02	. 13
Virginia City, Mont	30	5	?	140	14	15	70	85	------	65	1. 65	2½	1	14 x 18	180	. 35	. 02	. 37
Henderson No. 1, Mont	45	10	9, 000	200	40	0	600	600	4	270	2. 00	5	1	22 x 24	120	. 10	. 02	. 12
Henderson No. 2, Mont	15	2	3, 500	320	64	0	150	150	16	80	3. 20	2 and 4	1	22 x 24	120	. 06	. 02	. 08
Wisconsin Gulch, Mont	27	20	15, 000	283	13	1, 500	1, 300	2, 800	24	220	. 10	4, 5, and 6	2	44 x 40	1, 900	. 34	. 035	. 375
Stemwinder, Mont	70	5	18, 000	300	50	50	130	180	12	225	3. 34	3	1	24 x 18	240	. 07	. 02	. 09
Diamond City, Mont	12	10	?	350	70	800	100	900	16	125	. 61	2, 3, and 4	1	32 x 36	2, 700	. 08	. 02	. 10
Superior, Mont	60	5	67, 000	560	70	?	85	?	16	90	?	2½	1	48 x 60	5, 000	. 09	. 03	. 12
Hocken-Smith, Idaho	12	5	10, 000	80	40	------	------	150	8	75 150	1. 60	1¾ and 2	1	18 x 18	600	. 10	------	. 10

[1] Cubic yards gravel washed per day per miner's inch; where water only used part of day, duty is calculated on basis of hours used.

[2] Width by depth.

[3] 1½ to 3 feet gravel overlain by volcanic ash.

[4] 350 when washing.

[5] 175 when washing.

[6] Including 11 feet of top soil.

[7] 28 feet of clay over 12 feet of gravel.

[8] Overlain with 40 feet volcanic ash.

[9] Exclusive of ditch repairs, cleaning up, and supervision.

[10] Includes $0.02 for storage of tailings.

[11] 900 for actual days washing.

[12] To July 9.

TABLE 47.—*Operating data on hydraulic mines*—Continued

Mine and situation	Gravel					Water supply, miner's inches				Pressure water head, feet	Duty of total water, cubic yards	Nozzles		Sluices		Operating cost per cubic yard		
	Average thickness, feet	Boulders over 12 inches, percent	Volume washed, cubic yards			By wash water	Pressure water	Total water	Hours used per day			Diameter, inches	Number used at one time	Size, inches	Length, feet	Labor and supervision	Supplies	Total
			Season	Day	Per man-shift													
Golden Rule, Idaho	12	5	7,000	110	37	------	275	------	12	200	.80	4 and 3½	1	30 x ?	160	.095	.03	.125
Fortune, Colo	30	1	?	440	73	?	190	?	20	200	?	3 and 4	1	26 x 30	72	.065	.02	.085
Dodman, Colo	50	5	[13] 8,500	104	52	350	50	400	8	40	.78	3	1	28 x 18	132	.07	.01	.08
Redding Creek, Calif	9	8	56,600	540	25	------	1,200	1,200	24	300	.45	5	2	48 x ?	48	.155	.035	.19
Browning, Oreg	12	2	30,000	667	111	------	[14] 900	900	24	300	.74	4½ and 5	2	48 x ?	32	.045	.015	.06
Llano de Oro, Oreg	25	0	50,000	[15] 286	57	10,000	[16] 1,700	11,700	24	360	?	3¾	4	30 x 24	880	.07	.01	.08
Platurica, Oreg	35	1	100,000	500	55	0	[16] 650	650	24	450	.77	3	2	32 x ?	304	.07	.01	.08
Davis, Idaho	5	15	?	64	16	0	[16] 240	240	24	125	.51	1½	2	32 x 24	72	.25	.02	.27
Gallia, Calif	33	10	12,000	200	37	0	725	[17] 200	?	265	?	4 and 4½	3	24 x 24	125	.15	.02	.17
Lewis, Oreg	18	3	7,000	[18] 49	48	0	300	[19] 300	9	180	.93	3 and 4	2	30 x ?	95	.085	.015	.10
Connors, Oreg	7	1	?	18	6	0	28	28	8	300	1.93	⅝ or ¾	1	12 x 10	90	.67	.26	.93
Eldorado Bar, Mont	14	5	?	500	41	290	60	[20] 350	24	[21] 100	1.43	1¼ or 1½	1	30 x 36	400	.12	[22] .17	.29
Old Garden Gulch, Idaho	7	1	?	160	27	0	400	400	24	[21] 100	.40	2½ and 3½	2	30 x ?	96	.15	[23] .23	.38
Golden Rule, Idaho	10–20	?	6,930	110	37	?	?	?	?	200	?	4 and 3½	1	30 x ?	?	.095	.03	.125

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[13] To July 18.

[14] Includes water for Ruble elevator.

[15] 400 for actual days working.

[16] Includes water for hydraulic elevator.

[17] Including water for hydraulic elevator and generator.

[18] 106 for actual days washing.

[19] Including water for giant at Ruble hydraulic elevators.

[20] Water pumped 120 feet vertically.

[21] 40 pounds per square inch furnished by booster pump.

[22] Includes power for pumping $0.15.

[23] Includes power for pumping $0.21.

EXCAVATION BY TEAMS OR POWER-OPERATED EQUIPMENT OTHER THAN DREDGES

During the last decade conventional excavating machinery such as drag scrapers, power shovels, draglines, tractors, trucks, and bulldozers have been adapted to placer mining. Although the earlier adaptations often were commercially unsuccessful, a technique was developed rapidly that resulted in numerous successful operations. They are applicable to the mining of deposits too small to warrant capital expenditures for standard bucket dredges [63] or deposits otherwise unsuited for dredging and to deposits too low in grade to be worked profitably by small-scale hand methods and to those where there is an insufficient volume of water for ground sluicing and insufficient head for hydraulic mining. The excavating and transporting equipment is used with washing and gold-saving plants, which may be either stationary on land, mobile on land, or mounted on a barge or pontoons floating in a pond.

Gardner, Johnson, Allsman, and others have discussed the equipment, methods of operation, and costs in a number of Bureau of Mines publications [64] and numerous articles on the subject have appeared in the technical journals.

Teams and tractors.—Teams may be employed in small-scale work for the selective mining of thin gravels and may be used in shallow water where a tractor could not operate. The gravel usually is loosened by hand-picking or plowing and drag scrapers excavate and pull the material to the head of the sluice. In some cases barren soil over the deposit may be dragged to one side, uncovering the pay dirt. In others, where some water is available for ground sluicing, the overburden may be washed off after it has been loosened by plowing or harrowing. The gravel often is delivered to a trommel at the head of the sluice for disintegration, washing, and screening out the boulders or coarse particles before it passes to the sluices.

Tractors are used at a number of places instead of teams for pulling drag scrapers, greatly increasing the excavating and hauling capacity of the equipment and reducing the hand labor required for loosening the ground by picking.

Another method employs two- or three-drum clutched hoists for handling drag scrapers. The hoist is set at the receiving hopper or bin and by means of a pull rope on one drum attached to the front or bail of the scraper drags the scraper into the gravel and pulls the load to the bin. A tail or "back-haul" rope on the other drum passes through a sheave suitably anchored to a mast or deadman and thence to the back of the scraper and drags the empty scraper back to the pit. By moving the tail sheave, the scraper can be made to dig from all directions about the hoist. Various methods are employed for anchoring the tail sheave; for example, instead of a deadman or an anchor rope stretched between two deadmen, a heavily weighted car or truck may be used, which can be moved readily and quickly when it becomes necessary to change the line of drag. For digging in coarse, bouldery gravel or in hardpan, a hoe or open-bottom scraper is the most efficient type (considered purely as an excavator) but is liable to drop much of the gold en route to the washing plant. On the other hand, although a closed-bottom scraper will retain the excavated material better, it is an inefficient digging machine in coarse, rubbly material. Figure 124 shows a typical drag-scraper layout.

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[63] It is reported that a small bucket dredge can be purchased now at a cost as little as that for equipping the larger dragline plants.

[64] Works cited in footnote 56 (p. 325).

A closed-type bucket operating on a slackline cableway may be employed instead of a drag scraper. The bucket usually is similar to a toothed drag scraper but has a solid bottom and is hung below a carrier block by a swinging bail. The carrier block travels on a track cable between the loading and dumping points. One end of the track cable is anchored to a deadman or bridle cable and the other is attached to a tackle block on the end of a tension cable. The tension cable is reeved through blocks on a mast and then passes to the drum of a hoist. The carrier with bucket suspended below it runs by gravity on the inclined tightened track cable to a position above the loading point. By "slacking off" the track cable the bucket is lowered to the ground.

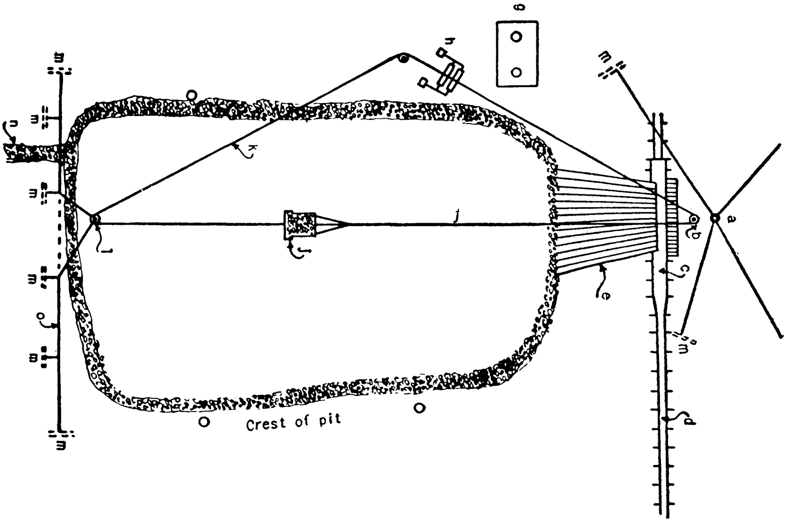

FIGURE 124.—Typical drag-scraper layout. *a*, Gin pole; *b*, head sheave; *c*, dump box; *d*, sluice; *e*, timber ramp; *f*, pull cable; *g*, boiler house; *h*, 2-drum hoist; *i*, pit; *j*, drag scraper; *k*, tail rope; *l*, tail sheave; *m*, deadmen; *n*, bedrock drain; *o*, 1-inch anchor cable.

A drag or inhaul cable, one end of which is attached to the front end of the bucket and the other to a second hoist drum, is then used to drag the bucket ahead until it has obtained a load. The tension cable is then tightened, which raises the track cable and with it the carrier and bucket, and the inhaul hoist is thrown into gear, carrier and bucket being pulled back to the dumping point near the mast. With this type of equipment, gold once in the bucket is retained until the dump is reached. In addition to the scraper and hoist, masts, and heavy overhead cables, a carrier and traveler block are required, increasing the first cost of equipment and the installation cost. Slackline cableways are not as flexible in operation and are not as readily moved to enable excavation of the entire area, which is often irregular in shape.

Table 48 gives some data on placer operations employing scrapers.

TABLE 48.—*Data on placer operations using scrapers*

Mine and situation	Gravel		Excavator			Haul		Washing plant		Operating cost per cubic yard					
	Depth, feet	Digging	Type	Bucket size, cubic feet	Power	By—	Distance, feet	Type	Gravel washed per hour, cubic yards	Excavation	Hauling	Washing and gold saving	Pumping	General	Total
Robbins, Utah	4	Easy	Team and slip scraper.	3.85	Burros	Team and slip scraper.	25	Screen and sluice.	0.8	$0.48	(1)	$0.46	$0.05		$0.99
Scott & Case, Nev.	4	do	do	?	Horses	Truck	[2]3.5	Shaking box and sluice.	2.0						1.50
Roberts, Calif.	(3)	Loose	Tractor and drag scraper.	6.75	Gasolene motor.	Tractor and scraper.	300	Grizzly and sluice.	2.5	.40	(1)	.17	(4)	$0.05	.62
Delaney, Wash.	6	Easy	Hoist and drag scraper.	5.4	do	Hoist and scraper.	20	do	3.8	.39	(1)	.17			.56
McElroy, B. C.	6	Medium	do	6.75	do	do	50	do	7.5	.16	(1)	.11	(4)	.03	.30
Mammoth Bar, Calif.	30	Easy	Scraper on headline.	27.0	Gasolene, 95-hp.	Headline and hoist.	300	do	12.0	.160	[5]$0.050	.045	.015	.095	.365
5 mines at Fairbanks, Alaska.	15–35		Hoists and drag scrapers.	47.25 to 67.50	Steam	Hoists and scrapers.		Sluices	25.0						.45–.90
1 mine, Fairbanks, Alaska.	16	(6)	do	54.0	do	Hositing car on incline.	300–400 (max.)	Sluices and undercurrents.							.59
Willow Creek, Alaska.	16	(7)	do	33.75	do	do		Sluices							.51
Innoko, Alaska	8	(8)	Slip scraper and hoist.	20.0	do	Scraping to boxes.		do	10.0						[9].87

[1] Included with excavation.
[2] Miles.
[3] Old tailings pile.
[4] Included with washing.
[5] Wages of general mechanic.
[6] Upper 5 feet hydraulicked off; 3 feet of upper gravel scraped to waste; 5 feet of gravel and 3 feet of schist bedrock scraped and sluiced.
[7] Nine feet of overburden ground-sluiced; 7 feet of gravel and soft bedrock scraped and sluiced.
[8] Four feet of overburden and upper gravel first removed by ground-sluicing for 14 cents per cubic yard; 4 feet of gravel and bedrock then scraped and sluiced.
[9] Average cost for entire 8 feet, ground sluicing and scraping. Scraping alone cost 1.55 cents per cubic yard.

Power shovels and dragline excavators.—Both power shovels and dragline excavators have been used successfully in placer mining during recent years, and each has its particular field. The digging element of the power shovel is a dipper attached to a rigid dipper stick, which, together with the hoist rope, thrusts the dipper into the gravel bank. The dragline excavator is similar in appearance but usually has a longer boom, and the digging element is a form of drag scraper having no dipper stick. The scraper is attached, front and back, by suitable bridles to a rope, one leg of which passes over a crown pulley on the end of the boom and thence to the hoist drum, and the other leg to the companion hoist drum. The dragline bucket then loads itself in the same manner as a drag scraper, but instead of the load being dragged to the discharge point it is hoisted to the required dump elevation and may be swung by revolving the boom or the entire machine, as the case may be.

Caterpillar mounting generally is employed for both shovel and dragline excavators of the full-revolving type. Dragline excavators can dig considerably below grade and under water; by means of the longer boom, they can deposit the gravel at a greater distance from the machine. This type of excavator, therefore, usually is employed when the washing and gold-saving plant is mounted on a float. The dragline works from the top of the bank, whereas the shovel works down in the pit. Bulldozers often are used for removal of soil overburden prior to mining the gravel and for cleaning up corners, pushing gravel up to the shovel, or leveling tailing piles.

Figure 125, *A* and *B*, shows typical arrangements using floating washing and gold-saving plants ("boats"). Figure 125, *C*, is a plan of an operation where the gravel is first dug by a power shovel and is then picked up by a dredge equipped with a light bucket line.

Where floating plants can be employed, operating costs usually are lower than with dry-land plants, largely because the disposal of tailings is simplified and is cheaper. Figure 126 shows the essential features of a floating washer and gold-saving plant. In operation, the excavator discharges the gravel into a hopper on the float, whence it is fed to a trommel, where it is washed and the boulders are removed and then to the sluices or other gold-saving devices. Table 49 presents data on placer mines employing power excavators with floating washing and gold-saving plants.

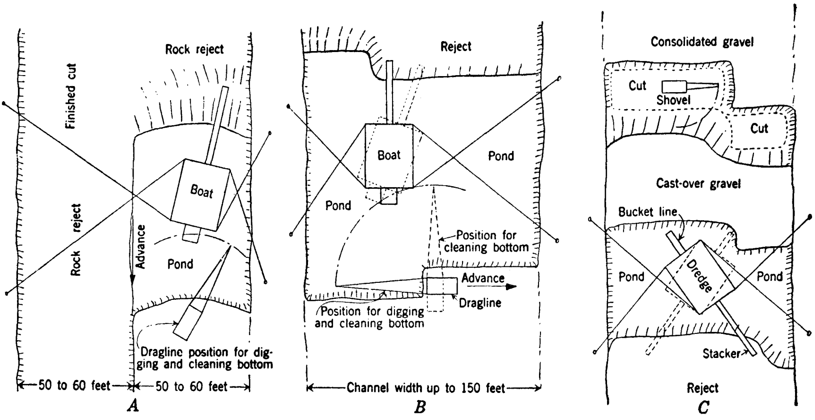
Rock reject
Finished cut
Boat
Rock reject
Advance
Pond
Dragline position for digging and cleaning bottom
50 to 60 feet
50 to 60 feet
A
Reject
Boat
Pond
Pond
Position for cleaning bottom
Advance
Dragline
Position for digging and cleaning bottom
Channel width up to 150 feet
B
Consolidated gravel
Cut
Shovel
Cut
Cast-over gravel
Bucket line
Dredge
Pond
Pond
Stacker
Reject
C

FIGURE 125.—Typical arrangements using floating washing plant.

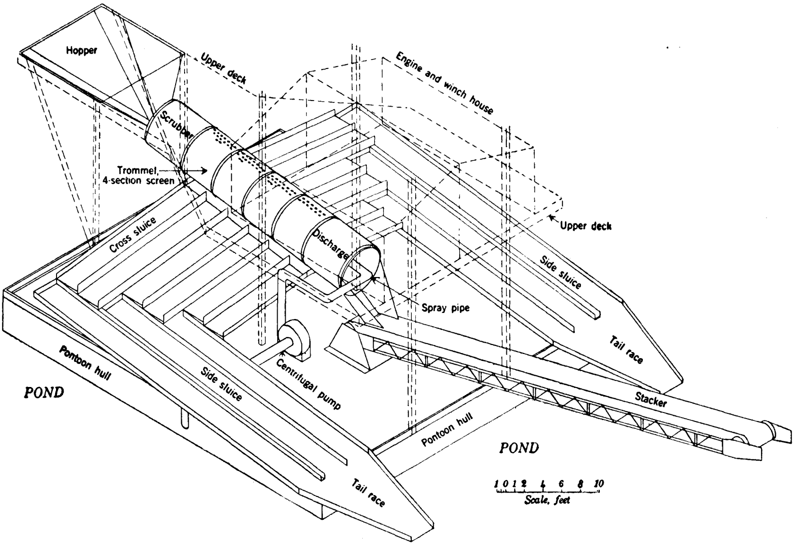
Hopper
Upper deck
Engine and winch house
Scrubber
Trommel, 4-section screen
Upper deck
Cross sluice
Discharge
Side sluice
Spray pipe
Tail race
Stacker
Centrifugal pump
Pontoon hull
POND
Side sluice
Pontoon hull
POND
Tail race
1 0 1 2 4 6 8 10
Scale, feet

FIGURE 126.—Essential features of a floating washer and gold-saving plant.

TABLE 49.—*Data on placer mines using power excavators with floating washing plants* [1]

Mine and situation	Gravel and overburden		Bedrock		Excavators				Washing and gold-saving plant				
	Average depth, feet	Digging	Kind	Depth mined, inches	Type	Bucket size, cubic feet	Boom length, feet	Power	Hull	Size of hull, feet	Draft, inches	Power	Gravel washed per hour, cubic yards
Lynx Creek, Ariz.	18	Easy	Caliche	6–18	Dragline (2)	40–47	61–60	Gasoline-electric, 75 and 175 hp.	Wooden	50 x 46 x 5	40	Electric, 135 hp.	125
Fay, Calif.	10–12	do	Granite	18–24	Dragline and bulldozer.	40	50	Diesel, 135 hp.	do	40 x 30 x 2.7	24	Diesel, 64 hp.	90
Consuelo, Calif.	3.5–8	do	Soft sandstone	None	Dragline	34	55	Diesel, 130 hp.	do	17 x 35 x 3	28	Diesel, 90 hp.	50
Richter, Calif.	12	do	Soft clay	Variable	do	34	40	Diesel, 75 hp.	do	30 x 30 x 4.5	42	Diesel, 60 hp.	79
Cinco Mineros, Calif.	12	do	Soft porphyry	12–14	Dragline and bulldozer.	40	50	Diesel, 130 hp.	do	32 x 42 x 3½	32	Diesel, 85 hp.	82
Penn, Calif.	10–20	Medium hard.	Greenstone	----------	do	61	80	Diesel, 125 hp.	do	30 x 40 x 3	24	Diesel, 65 hp.	104
England,[7] Calif.	14 (4–25)	Easy	Lava ash	----------	Dragline	34	60	Diesel, 102 hp.	do	----------	28	Diesel, 25 hp.	127
Midland, Calif.	9	do	Sandy hardpan	6–8	do	34	45	Diesel	do	25 x 40 x 3	24	Diesel, 50 hp.	122
Pioneer No. 3, Calif.	7	do	Clay	6	Dragline and bulldozer.	81	65	Electric, 125 hp.	Steel	32 x 35 x 3½	30	Electric, 223 hp.	180
Carlson-Sandburg, Calif.	13	Medium	do	----------	do	34	50	Diesel, 130 hp.	Wooden	34 x 42 x 3½	22	Electric, 57.5 hp.	130
Olson, Calif.	6½	Easy	do	6	do	40	50	Diesel, 95 hp.	----------	30 x 40 x 3½	40	Diesel, 50 hp.	167
Collins, Calif.	11	do	Porphyry	18	do	40	50	Diesel, 120 hp.	----------	30 x 40 x 3½	26	Diesel, 100 hp.	100
Folsom, Calif.	9	Hard	----------	----------	Shovel [11]	27	----------	Gasoline, 85 hp.	----------	28 x 38 x 3	20	Electric, 51.5 hp.	90
Lilly, Calif.	8	Medium tight.	Soft clay	12	Dragline (2)	54–67	50	Diesel, 123 hp.	Steel	40 x 36 x 3	30	Electric, 102.5 hp.	125
Schwegler, Calif.	4.5–12	do	Tuff	6	Dragline	40	50	Electric, 121 hp.	Wooden	40 x 35 x 3	26	Electric, 125 hp.	100
Cooley, Colo.	17.5	----------	Decomposed granite.	6–18	do	40	45	Diesel, 80 hp.	----------	40 x 34 x ?	----------	Diesel, 95 hp.	60
Fairplay, Colo.	[15] 21	Easy	Granite	----------	do	67	70	Gasoline, 215 hp.	Wooden	28 x 48 x 2½	18	Gasoline, 144 hp.	50

Prickly Pear, Mont	8-9	do	do		do	67	70	Diesel, 120 hp.	do	28 x 40 x ?		Gasoline, 150 hp.	100
A.[17]	9.5	Hard	Decomposed granite.		do	34	50	Diesel					
B.[17]	6	Easy			do	40	50	do					

Kind	Operating time per day, hours	Men employed per day				Gravel dug and washed, man-hours per cubic yard	Water consumed, miner's inches	Cost of plant and equipment	Operating cost per cubic yard			
		Excavation	Washing	General	Total				Excavation	Washing	General and miscellaneous	Total
Lynx Creek	18.0	12	12	6	30	0.106						[2] $0.10
Fay	15.25	4	3		7	.041	35	$37,500				.11
Consuelo	20.5	6	5		11	.086	40	47,000				[3] .12
Richter	19.0	6	5	1	12	.065		[4] 27,000				[4] .12
Cinco Mineros	19.5	7	5	1	13	.065	50	34,500	$0.040	$0.056	$0.015	[5] .111
Penn		6	4	4	14			[6] 100,000	.033	.038	.021	.092
England	15.7	6	5	1	12	.048	[8] 570	40,000	.033	.037	.038	.108
Midland	15.3	2	3	6	11	.053	[8] 1,400	40,000				.150
Pioneer No. 3	16.5	7	7	1	15	.044		60,000				[9] .090
Carlson-Sandburg	19	7	5	3	15	.040	100	?	.030	.030	.030	[10] .090
Olson	18	7	4	2	13	.035	40	43,600				[10] .090
Collins	19	5	6	5	16	.084	100	52,800				[10] .110
Folsom	15	2	5		7	.041	20	[12] 10,000	.0275	.0275		[13] .055
Lilly	21.5	7	5		12	.035		50,000	.040	.050	.010	[13] .100
Schwegler	19	6	5	1	12	.051	50	38,500				.125
Cooley	20	6	5	1	12	.080	55					[14] .18
Fairplay	12	2	5		7	.117						.25
Prickly Pear	20	6	10	1	17	.068	150		.150	.150		[16] .30
A									.037	.067	.014	.118
B									.0346	.0240	.0067	.0653

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[1] Data abstracted from Inf. Circ. 7013, except where otherwise noted.

[2] Includes all labor, $0.05; power, $0.02; and other costs, including depreciation, $0.02; but excludes royalties.

[3] Includes depreciation over a 4-year period but not owner's wages.

[4] Excludes wages of owner and two sons.

[5] Does not include depreciation on equipment.

[6] Includes cost of two light draglines that could not be used.

[7] Magee, James F., A Successful Dragline Dredge: Trans. Am. Inst. Min. and Met. Eng., vol. 126, 1937, pp. 180–196.

[8] Gallons per cubic yard.

[9] Based on handling 4,000 cubic yards per day, including depreciation.

[10] Includes depreciation but not royalties.

[11] Gravel is dug and cast aside with gasoline shovel and is picked up and washed by a light bucket-line floating dredge.

[12] Boat only.

[13] Owner's wages not included.

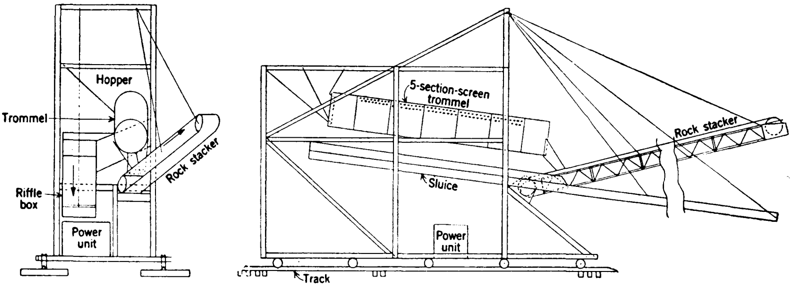
[14] Estimated for first 6 weeks' operation and includes preliminary nonrecurring expenditures.

[15] 12 feet gravel left unmined on bedrock as it was too deep to handle with dragline.

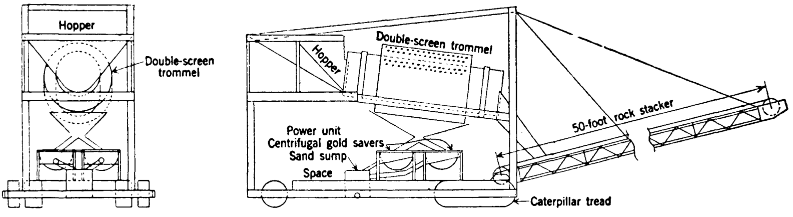
[16] Includes 5 to 7 cents depreciation.

[17] Magee, James F., Dragline Dredging: Trans. Canadian Inst. Min. and Met., vol. 40, 1937, pp. 117–125.

Land plants are of varied design and are not standardized. They are of two general types, however—movable and stationary. Movable plants may be mounted on caterpillars, on a wheeled chassis running on tracks, or on wide, flat wheels that may be self-propelled or may be towed by the excavating machine. A semiportable type is mounted on skids. When movable plants are used, the disposal of tailings is facilitated, as the plant follows the excavator and can discharge the tailings into the worked-out part of the pit. With stationary plants it is necessary to provide auxiliary equipment for removing and disposing of the tailings, introducing an additional item of operating expense. The essential elements of a movable washing and gold-saving plant (fig. 127) are the chassis and means of locomotion and framework to support the equipment. The equipment consists of a washer, which is usually a trommel screen, power unit (for operating pumps), pumps, gold-saving equipment, and rock stacker. The gold-saving equipment may consist of riffled sluiceboxes, power rockers, centrifugal gold savers, or similar equipment. Figure 128 illustrates two general plans of operation using movable dry-land washing plants.


Hopper
Trommel
Rock stacker
Riffle box
Power unit
5-section-screen trommel
Rock stacker
Sluice
Power unit
Track

A-TROMMEL AND RIFFLE SLUICE


Hopper
Double-screen trommel
Double-screen trommel
Hopper
Power unit
Centrifugal gold savers
Sand sump
Space
50-foot rock stacker
Caterpillar tread

B-TROMMEL AND CENTRIFUGAL BOWLS

FIGURE 127.—Essential elements of movable dry-land washing and gold-saving plant.

Stationary land plants are similar to movable land plants with respect to washing equipment and differ from them mainly in their lack of mobility and in the method of disposing of the tailings. Usually they are employed where water is scarce and frequently require a pumping plant for delivering wash water through pipe lines from wells or other sources of supply. They can be of more rugged construction than movable plants, are cheaper to build, and can be so situated as to take advantage of natural sites suitable for stacking the tailings. They have the additional advantage that wash water can be kept out of the pit where the excavator is working and the bedrock is thus exposed and can be cleaned more easily and thoroughly. Because of the high-operating cost, their use is restricted to mining of higher-

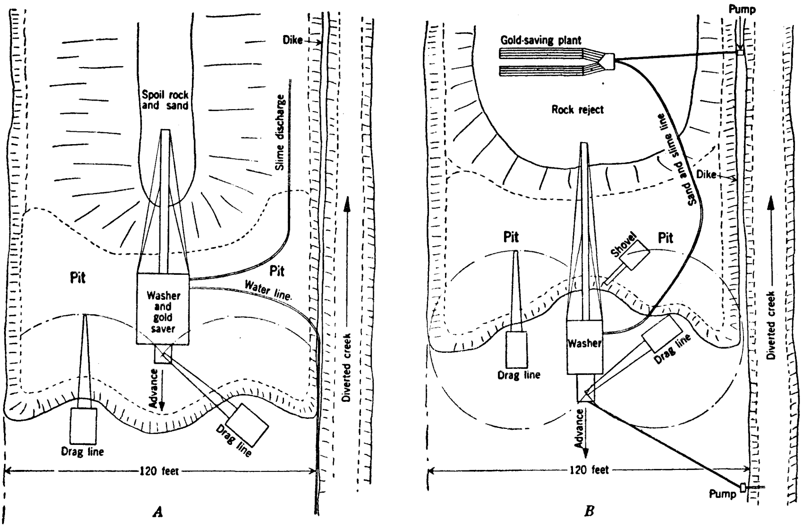

Dike
Spoil rock and sand
Slime discharge
Pit
Pit
Water line
Washer and gold saver
Advance
Drag line
Drag line
Diverted creek
120 feet
A
Pump
Gold-saving plant
Rock reject
Sand and slime line
Dike
Pit
Shovel
Pit
Washer
Drag line
Drag line
Advance
Diverted creek
120 feet
Pump
B

FIGURE 128.—Typical set-ups for movable, dry-land washers.

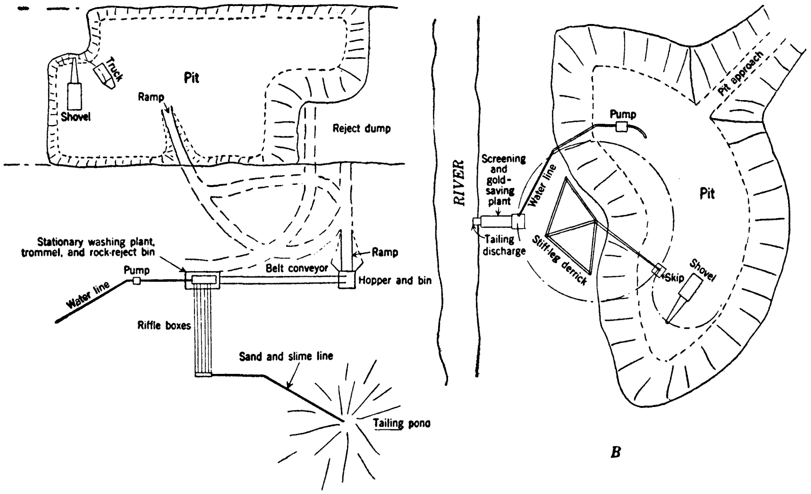

Truck
Pit
Shovel
Ramp
Reject dump
Stationary washing plant, trommel, and rock-reject bin
Pump
Water line
Belt conveyor
Ramp
Hopper and bin
Riffle boxes
Sand and slime line
Tailing pono
RIVER
Screening and gold-saving plant
Water line
Tailing discharge
Stiff-leg derrick
Pump
Pit approach
Pit
Skip
Shovel
B

FIGURE 129.—Typical set-ups for stationary washing plants.

grade deposits. Figure 129 shows typical set-ups for stationary plants. Table 50 presents data on placer operations employing dry-land plants.

TABLE 50.—*Data on placer mines using power excavators with dry-land washing plants*

Mine and situation	Gravel and overburden			Bedrock		Excavators				Hauling gravel		Washing and gold-saving plant		
	Average depth, feet		Digging	Kind	Depth mined, inches	Type	Bucket size, cubic feet	Boom length, feet	Power	By—	Distance, feet	Mounting	Equipment	Power
	Stripping	Gravel												
Skeels, Calif	0	8–12	Easy	Granite	8	Shovel	34		Gasoline	None		Movable steel frame on caterpillars.	Trommel, 4 centrifugal bowls.	Diesel, 44 hp.
Pantle, Calif	10–12	18–20	do	do	24	Dragline	27	50	Diesel, 85 hp	do		do	do	Electric, 38 hp.
Pan-Due, Colo	0	20	do	do	24	Shovel	34	22	Gasoline, 100 hp.	do		Movable steel frame on R. R. car trucks.	do	Gasoline, 80 hp.
Fairplay, Colo	3	?	do	?	?	Dragline	34	50	Diesel	do		Movable steel frame on wheels.	Trommel and sluice.	Gasoline.
Blackhawk, Colo	0	26	do		12–15	Shovel	34	23	Gasoline, 150 hp.	do		Movable, steel on dragline chassis.	Trommel and sluice, and Wilfley table.	Electric.
Humphreys, Colo		15	do	Granite schist.	1+	2 draglines 1 shovel	67 34	50 22	do Diesel, 130 hp	do		Movable, on caterpillar-tractor chassis.	Trommel, sluices corduroy tables.	Gasoline, 480 hp.
Dixie, Idaho	10	13	do	do	6–12	Dragline	20	45	Gasoline	do		Movable, on wheels on timber track.	Trommel sluices.	Gasoline, 34 hp.
Humphreys, Mont.[8]	0	17	do	Volcanic ash.	6	2 draglines 1 shovel	67 47	50 ?	Electric, 100 hp. Electric, 75 hp.	do		Movable, on 3-tractor chassis with caterpillar treads.	do	Electric, 710 hp.
Eldorado, Mont	12–13	15	do	Shale and clay.		Dragline	30			do		Movable, on 8 wheels.	do	Gasoline, 24.5 hp.
Hallett, N. Mex	2–6	10				do	34		Fuel oil	do		Movable, on wheels on wooden stringers.	Trommel and 4 centrifugal bowls.	Gasoline, 65 hp.
Jett-Ross, Wyo	2–12	2–8	Loose to tight.	Schist, diorite.	24–60	do	34	50	Oil burner, 125 hp.	0		Movable, wood frame on skids.	Grizzly and sluices.	
Fisher, Wyo	3	10	Easy	do	18–24	do	34	60	Fuel oil	0		Movable, wheels on rails.	Trommel and sluice.	Gasoline, 100 hp.

Ash Canyon, Ariz	8	10	Hard	Conglomerate.	2–3	Shovel	13.5		Gasoline, 65 hp.	Truck	1,500	Stationary wooden frame.	Trommel, 1 centrifugal bowl.	4-cylinder gasoline engine.
Chittendon, Calif	0	15	do	Granite	6–10	do	13.5	35	Gasoline	Two 3-cubic-yard trucks.	400	do	Trommel sluices.	Do.
Von der Hellen, Calif	0–12	40	do	Serpentine.	18–24	do	34	21	do	5 trucks	90	do	do	Gasoline.
Guerin, Calif	0	27	do		None	do	10		do	Derrick	[13] 35	do	do	Do.
La Grange, Calif	0	25	do	Tuff	Few	do	20		do	Trucks	600	Steel frame	do	Electric, 60 hp.
Rhyolite, Nev	10	4	Easy	Gravel		do	17			do	5,000	Wooden frame	do	Gasoline.

Mine and situation	Washing saving	and gold-plants	Men employed, number per day					Gravel dug and washed, man-hours per cubic yard	Water, gallons per cubic yard	Cost of plant and equipment	Operating cost per cubic yard			
	Gravel washed per hour, cubic yards	Operating time per day, hours	Excavation	Hauling	Washing	General and miscellaneous	Total				Labor and supervision	Power and supplies	General	Total
Skeels, Calif	40	12.8	2	0	5	0	7	0.109	670		$0.063	$0.097		[1] $0.160
Pantle, Calif	[2] 100	16.0	2	0	5	0	7	.090		[3] $12,000				[1] [4].080
Pan-Due, Colo	50	14.0	2	0	7	0	9	.103	480					[1].200
Fairplay, Colo	20	?	2	0	4	3	9				.200	.160	.100	.460
Blackhawk, Colo	70	8.0	2	0	9	5	16	.286	1,100					[5].250
Humphreys, Colo	180	15.0	8	0	15	3	26	.096	580	147,000				.250
Dixie, Idaho	?	10.0	3	0	2	2	7	?	[6] 600		.032	.032	.011	[7].075
Humphreys, Mont.[8]	240	20.0	18	0	9	5	32	.053	1,400	264,795	.055	.057	.006	[1].118
Eldorado, Mont			2	0	6	1	9	?	[6] 420					.110
Hallett, N. Mex	37		4	0	5	1	10	?		[9] 24,480				[1].250
Jett-Ross, Wyo	75	20	6	0	8	0	14	.075	3,200	27,000				[10].085
Fisher, Wyo	73	?	6	0	11	0	17	?	450	27,200	.045	.045	.029	[10].119
Ash Canyon, Ariz	4.5	7	1	3			4	[11] 1.060	670					[4].130 [11].700
Chittendon, Calif	40	9	1	2		2	5	.139	1,200					[12].150
Von der Hellen, Calif	75	16	4	10	8	6	28	.210	2,000		.150	.180	.020	.350
Guerin, Calif	7	7	1	0	1		2	.367		[14] 2,500				[14].250
La Grange, Calif	62	8	1	2	2		5	.080	1,800	30,000	.066	.044	.052	[10].162
Rhyolite, Nev	10	12					5	.500	1,600		.250	.230	.280	[15].760

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[1] Excluding depreciation and royalties.

[2] Includes stripping.

[3] Washing and gold-saving plant only.

[4] Based on total yardage (stripping and gravel).

[5] When running 40 percent of time.

[6] Gallons per minute.

[7] Excluding income tax, depreciation, and royalties.

[8] Corry, Arthur V., Dragline Installation for Recovering Gold at Virginia City, Mont.: Min. and Met., October 1936, pp. 467–470.

[9] Not including dragline.

[10] After depreciation.

[11] Based on yardage washed.

[12] Excluding time of 2 owners, repairs and depreciation.

[13] Vertical lift.

[14] Excluding wages of 2 owners.

[15] Includes plant rental, stripping cost, and royalties.

In 1936, 66 percent of the total gold recovered from [illegible] United States (Philippine Islands and Puerto Rico excluded) [illegible] produced by floating dredges of the connected-bucket-ladder type. In 1935 the percentage was 67.11.[65] In 1936 there were 38 dredges operating in Alaska, 45 in California, 12 in Idaho, 5 in Oregon, 2 in Montana, and 1 in Colorado, a total of 103. These dredges recovered in that year 605,307 fine ounces of gold.

Earlier publications of the Bureau of Mines discuss gold dredging in detail.[66]

Dredging is adapted to large, thick deposits containing enough pay gravel (usually low-grade) to warrant the large initial investment required for testing the ground and for purchasing and erecting dredging equipment. Character of gravel and bedrock and the topography must be suitable, and enough water must be available to float the dredge and wash the gravel.

Dredges are not suitable for mining in canyons or narrow valleys where flood waters are likely to ensue after heavy rains or following rapid melting of snow, since such occurrences may result in serious damage to the equipment or in its total loss. Gravel containing many large boulders also is unsuited to dredge operation. The gravel should be tight enough to hold water in the pond without excessive seepage. The most successful dredging operations have usually been in the wide, lower river valleys.

The essential parts of a dredge comprise a hull, a digging-ladder assembly with bucket line and tumblers, screen, sluice boxes and riffles or tables (and sometimes other gold-saving equipment, such as jigs), a stacker for disposal of the screen oversize, pumps, lines and spuds to hold the boat in position while digging and for moving up the dredge, and the necessary power equipment and drive engines or motors. Figure 130 shows the Lancha Plana dredge, El Oro No. 3 mine,[67] in plan and elevation.

The buckets range in capacity from about 2½ or 3 cubic feet each on the smaller dredges to 18 cubic feet on the largest. During 1932 the average depth of gravel dredged at a number of operations was 35 feet, and dredges are in operation that dig 82 feet below water; a dredge having a digging depth of 110 feet and an estimated capacity of 15,000 cubic yards per day was built in 1934.[68] Late in 1938 a new dredge was under construction [69] in the Hammonton field, California, equipped with 18-cubic-foot buckets and designed to dig 115 feet below water line against a 40-foot bank. The hull, 80 by 250 feet, weighs 1,800 tons.

Table 51 gives data on a number of dredges.

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[65] Henderson, Chas. W., and Dunlop, J. P., Gold and Silver: Bureau of Mines Minerals Yearbook 1937, p. 136.

[66] Jennings, Hennen, The History and Development of Gold Dredging in Montana: Bureau of Mines Bull. 121, 1916, 63 pp.; Janin, Charles, Gold Dredging in the United States: Bureau of Mines Bull. 127, 1918, 226 pp.; Ash, S. H., Safety Practices in California Gold Dredging: Bureau of Mines Bull. 352, 1932, 31 pp.; Gardner, E. D., and Johnson, C. H., Placer Mining in the Western United States. Part III. Dredging and Other Forms of Mechanical Handling of Gravel, and Drift Mining: Bureau of Mines Inf. Circ. 6788, 1935, 81 pp.

[67] Patmon, Charles G., Methods and Costs of Dredging Auriferous Gravels at Lancha Plana, Amador County, Calif.: Bureau of Mines Inf. Circ. 6659, 1932, 16 pp.

[68] The Mining Jour., Arizona, Feb. 28, 1934, p. 19.

[69] Eng. and Min. Jour., September 1938, p. 62.

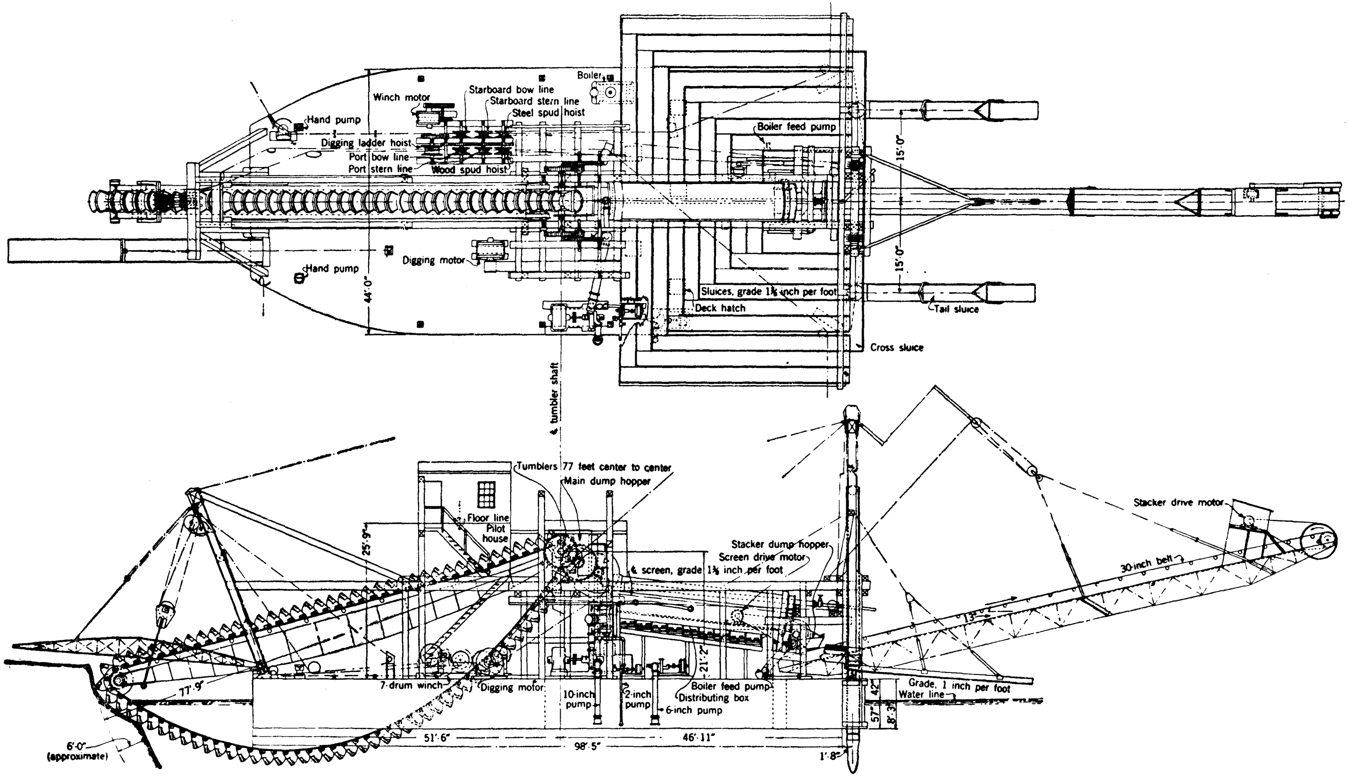
Boiler
Winch motor
Starboard bow line
Starboard stern line
Steel spud hoist
Hand pump
Boiler feed pump
Digging ladder hoist
Port bow line
Port stern line
Wood spud hoist
15'0"
15'0"
Digging motor
Hand pump
44'0"
Sluices, grade 1½ inch per foot
Deck hatch
Tail sluice
Cross sluice
℄ tumbler shaft
Tumblers 77 feet center to center
Main dump hopper
Floor line
Pilot house
25'9"
Stacker dump hopper
Screen drive motor
℄ screen, grade 1½ inch per foot
Stacker drive motor
30-inch belt
13°-2'
77'-9"
21'2"
7-drum winch
Digging motor
10-inch pump
2-inch pump
6-inch pump
Boiler feed pump
Distributing box
Grade, 1 inch per foot
Water line
42"
57"
8'3"
6'0" (approximate)
51'6"
98'5"
46'11"
1'8"

FIGURE 130.—Lancha Plana dredge, El Oro No. 3 mine.

TABLE 51.—*Data on typical gold dredges*

Dredging company and situation	Hull					Digging ladder			Screens, length × diameter, feet	Gold saving		Water pumped on dredge, gallons per minute	p on
	Construction	Dimensions, feet				Maximum digging depth, feet [1]	Size of buckets, cubic feet	Number of buckets in line		Total table area, square feet	Slope of tables, inches per foot		
		Length	Width	Depth	Draft								
Capital, Calif	Steel					70	18						
La Grange, Calif		104	44.5	9.5		30	10	60	36.5×6	2,000	1.25	7,500	
Lancha Plana, Calif	Wood	98.5	41	7	5	54	6	82	33 ×6	2,000	1.375	3,500	
Natomas No. 2, Calif	do					35	8.5	78					
No. 4	Steel					40	15	67					
No. 5	Wood					60	11	101					
No. 7	Steel					60	9	98					
No. 8	do					60	15	83		5,000			
Placer Development Co., Calif	Wood	100	43	9	8	38	7	72	30.5×6		1.5	6,200	370
Shasta Butte, Calif	Steel					41.5	7.5	73	30 ×6	1,300			440
Trinity, Calif	Wood	107	48	7.3	6		11	[3] 42	15 ×5	[4] 360	.75	7,500	400
Yuba, Calif.:													
No. 14						79	14		35 ×9	9,000			850
No. 15						82	14						850
No. 19						69	14						850
Continental, Colo	Wood	133	40	10	8	65	7	95	40 ×6	1,300	1.5	4,700	400
American, Idaho	do	80	40		4	18	2.5	46	30 ×4.67	1,000	1.0		130
Crooked River, Idaho	do	70	30		5	35	3	65	25 ×4.5	870		4,000	[5] 170
Idaho, Idaho	do			7	5	32	3.5	75	18 ×5	2,000	1.5		[6] 180
Empire, Oreg	do		40		5	36	6	77	36 ×6	1,300	1.25		295
Fairbanks Exploration Co., Alaska:													
No. 2	Steel	128	60	12	9.5	[7] 48	10	93	44.6×8	4,535	1.25 and 1⅛	5,500	749
No. 3	do	148	60	12	9.0	[7] 60	10	104	44.6×8	4,535	1.25 and 1⅛	5,500	749
No. 5	do	108	60	9	6.8	[7] 36	6	78	43.5×6	2,125	1.25 and 1⅛	4,000	489
No. 6	do	108	60	9	6.4	[7] 36	6	78	43.5×6	2,125	1.25 and 1⅛	4,000	489
No. 8	do	99	50	10.5	7.7	[7] 28	6	68	36.2×6	1,460	1.25 and 1⅛	4,000	424
Fisher & Baumhoff, Idaho (2 dredges)	Wood						4	75	30 ×5	(8)	(8)		365
Fisher & Baumhoff, Granite Creek, Idaho.	Steel						2.5	84	28 ×5	Jigs and sluices			163
Fisher & Baumhoff, Centerville, Idaho	do						6	76	28 ×5	Jigs and sluices			320

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[1] Below water.
[2] Electric power unless otherwise noted.
[3] Open-connected line.
[4] 90 feet of 4-foot sluice lined with angle-iron riffles.
[5] Diesel engine.
[6] Steam engines.
[7] Depth under water actually dug.
[8] 12 Pan-American jigs in series with two 16-inch by 42-foot sluices.

Because of their large capacity and the relatively small amount of labor required for operation, very low-grade gravel can be worked profitably under conditions favorable for dredge operation.

Cost of dredges.—The following statement regarding the cost of dredges is taken from a letter to the Bureau of Mines by L. D. Hopfield, department manager of the Natomas Co.:

> Dredges are built in various sizes from 2½ to 18 cubic feet of bucket capacity and designed to dig from 12 to 80 feet below water level. The cost ranges from $50,000 to $500,000 depending upon the capacity and design.
>
> A dredge with 6½-cubic-foot buckets, recently built in California, with steel hull and wooden superstructure, designed to dig 25 feet below water level, cost approximately $140,000. A 10-cubic-foot-capacity dredge designed to dig 40 feet below water level costs from $250,000 to $275,000. A 15- or 18-cubic-foot capacity dredge that will dig 60 to 80 feet below water level is estimated to cost between $450,000 and $500,000. These figures do not include the cost of land, camp buildings, roads, or freight.

A 200-ton steel hull for a 7½-cubic-foot dredge cost $19,000 in 1929. The 9-cubic-foot, flume-type Madrona dredge in Trinity County, Calif., was reported to cost $150,000, being made partly from salvaged material.[70]

Operating costs.—Table 52 presents operating data, including costs, on a number of gold-dredging operations.

According to Gardner and Johnson,[71] the average operating cost per cubic yard (taxes, insurance, and capital charges excluded) for 75 percent of the California yardage and 50 percent of the yardage of the other States in 1931 was 5.1 cents; the range was from 4 to 12 cents. These figures are based on a total of 36,000,000 cubic yards.

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[70] Eng. and Min. Jour., vol. 124, July 9, 1927, p. 62.

[71] Gardner, E. D., and Johnson, C. H., Work cited in footnote 66 (p. 350).

TABLE 52.—*Operating data on gold-dredging operations*

Dredge and year	Gravel		Average monthly yardage handled	Average operating time per day		Man-hours labor per cubic yard	Power consumption. kw.-hr. per cubic yard	Operating cost per cubic yard						
	Nature	Average depth, feet		Hours	Minutes			Labor and supervision	Power	Supplies	General operating expense	Prospecting	Overhead [1]	Total
La Grange, Calif., 1932	Loose, no clay, 5 percent boulders.	30	217,000	20	40		0.80	$0.018	$0.007	$0.010	$0.002	(2)	[3]$0.001	$0.038
Lancha Plana, Calif	Medium tight, 20 percent clay and loam.	40	63,500	18	45	0.042	1.88	.051	.020	.042	.011	(2)	.023	.147
Natomas, Calif, 1932:														
No. 2		30	152,000	20	40			[4].016	.013	.017	[4].007	(2)	(2)	.053
No. 4		28	271,000	20	29			[4].011	.010	.013	[4].004	(2)	(2)	.038
No. 5		50	147,000	18	32			[4].020	.015	.031	[4].007	(2)	(2)	.073
No. 7		36	177,000	20	32			[4].016	.014	.018	[4].005	(2)	(2)	.053
No. 8		58	237,000	20	25			[4].014	.014	.019	[4].005	(2)	(2)	.052
No. 10		51	259,000	20	27			[4].011	.013	.017	[4].004	(2)	(2)	.045
Total and average			1,243,000	20	11			[4].014	.013	.018	[4].005	(2)	(2)	.050
Placer Development Co., Calif., 1932	Loose, few boulders, little clay.	35	100,000	20	40	.036		.029	.014	.032	.0003	(2)	[3].0044	.0797
Shasta Butte, Calif, 1932	Fair digging, some clay and hard gravel.		130,000	18	00	.024		[4].019	.010	.008	[4].009	$0.008	(4)	.062
Trinity, Calif., 1932		25	100,000	20	30	.031	1.5							[5].060
Yuba, Nos. 14, 15, 19, Calif., 1932		54, 58, 74	900,000											[6].041
Continental, Colo., 1932	Some boulders and clay.		122,000	20	00	.027	1.1	.022	.019	.018	.0043	.0013	[3].0024	.067
American, Idaho, 1932	No boulders or clay	15	50,000	20	00	.039	.7	.040	.010	0.020				.070
Idaho, Idaho, 1932		20	75,000	22	00	.038		.045	.015	.040	(7)	(2)	(2)	.100
Empire, Oreg., 1932		24	75,000	20	30	.042								.062
Fisher & Baumhoff, Warren, Idaho, 1937.	Few boulders [8]	20	[9] 120,000	21	00			.014	.008	.017	.006	(2)	(10)	[11].045
Fisher & Baumhoff, Granite Creek, Idaho, 1937.	80-85 percent sand and fine gravel.	23	42,500					.0600	.0065	.0199	.0166	(2)	.0385	.1415
Fisher & Baumhoff, Centerville, Idaho, 1937.	do	23	133,000					.0143	.0075	.0166	.0012	(2)	.0362	.0758

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[1] Taxes, insurance, and deferred capital charges.
[2] Excluded.
[3] Taxes and insurance only.
[4] Supervision included with "General."
[5] Excluding only depreciation and depletion.
[6] Excluding taxes, insurance, and administrative expense.
[7] Included with labor.
[8] 15 to 20 feet of top soil on one side of valley.
[9] 2 dredges.
[10] General overhead including salaries and taxes included with "General operating expense."
[11] Depreciation and depletion 2 to 2½ cents additional.

According to Janin,[72] similar costs in 1914, based on 70,000,000 cubic yards, ranged from 2.65 to 5.3 cents and averaged 4.2 cents (wages were about 30 percent higher in 1931 than in 1914).

Costs in Alaska are much higher due to higher wage rates and costs for transportation of equipment and supplies, the short dredging season, and the cost of supplying water for dredging and cold-water thawing. According to Wimmler,[73] operating costs (exclusive of depreciation, depletion, royalty, interest, and other capital charges) are 15 to 35 cents per cubic yard. The following is abstracted from deductions made by Wimmler from incomplete data supplied by operators of 11 dredges running under a wide range of conditions on Seward Peninsula in 1921. The 11 dredges dug 1,323,500 cubic yards, 740,000 cubic yards of which was handled by 3 dredges. The operating costs were 15 to 38 cents per cubic yard, averaging 21.6 cents, and the total amount of invested capital was about $590,000. The low capital cost was due to the fact that a number of the dredges were second-hand and acquired for nominal sums. Depreciation on equipment averaged 4.64 cents per yard and interest at 6 percent, 2.67 cents, a total of 7.31 cents. The estimate does not include cost of land royalties. The number of working days per dredge was 75 to 129 for the season and averaged about 100 days.

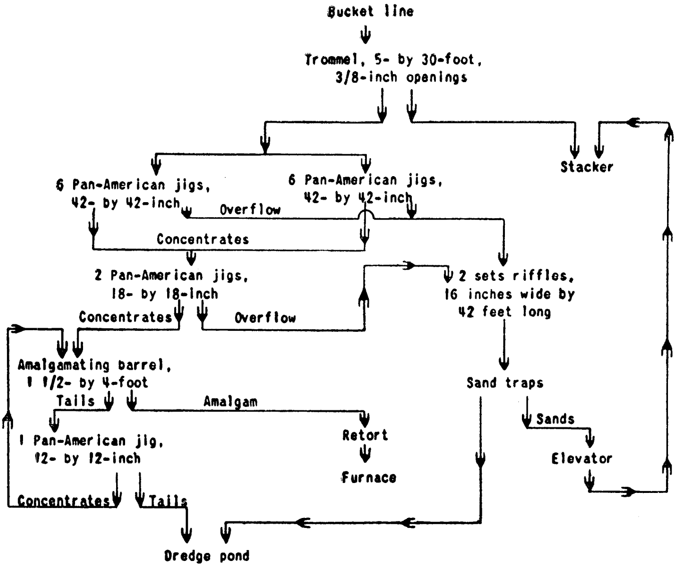
Bucket line
Trommel, 5- by 30-foot, 3/8-inch openings
Stacker
6 Pan-American jigs, 42- by 42-inch
Overflow
6 Pan-American jigs, 42- by 42-inch
Concentrates
2 Pan-American jigs, 18- by 18-inch
2 sets riffles, 16 inches wide by 42 feet long
Concentrates
Overflow
Amalgamating barrel, 1 1/2- by 4-foot
Sand traps
Tails
Amalgam
Sands
1 Pan-American jig, 12- by 12-inch
Retort
Elevator
Furnace
Concentrates
Tails
Dredge pond

FIGURE 131.—Flow sheet of Warren Creek dredge, Idaho, using jigs.

No recent data on dredging costs in Alaska are available to the authors.

Gold saving.—Most of the dredges in the Western States use riffled tables or sluices for saving the gold; some employ jigs, also (see flow sheet, fig. 131). Quicksilver is almost always used to collect the gold in the upper end of the table circuit. Clean-ups usually are made periodically—once a week or 10 days—dredging then ceases and the idle time is used to good advantage for making necessary repairs.

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[72] Janin, Charles, Work cited in footnote 56, p. 325.

[73] Wimmler, Norman L., Placer-Mining Methods and Costs in Alaska: Bureau of Mines Bull. 259, 1927, 236 pp.

Experienced dredge operators estimate that 60 to 90 percent of the gold contained in the deposits is recovered, although until recently, at least, it is doubtful if many estimates covering long periods have been made on the basis of careful sampling of the tailing. Excavating is done under water, and unless care is taken to dig well into the fractured and disintegrated bedrock, much gold will be lost there. Clay is troublesome in dredging and if present in the gravel in considerable quantities may carry a good deal of gold into the tailing. The use of jigs for saving fine or rusty gold that will not amalgamate readily has increased rapidly in recent years. They also serve a useful purpose when placed ahead of the sluices in removing heavy black sands that tend to pack behind and blind the riffles.

A discussion by P. Malozemoff of jigging as applied to gold dredging was published recently.[74]

Suction dredges with or without cutter attachments have failed repeatedly to be successful in raising more than a small percentage of the gold in placer deposits, although they have a large volumetric capacity in favorable material. (A suction dredge for Panama Canal service is reported to have a capacity of 500 to 1,000 cubic yards per hour.)

OPEN-PIT MINING

Open-pit methods are applicable to mining ore deposits that apex at or near the surface. If the deposit apexes below the surface, the overburden and barren capping overlying the ore must be removed in advance of open-pit mining. The removal of this material is known as stripping.

The stripping-pit limits must be extended beyond the limits of the ore pit to provide a bench, and the pit-wall slopes must be such as to prevent sloughing of overburden onto the ore area. Where the ore is to be hauled from the pit by locomotive or trucks, additional excavation is required to provide an approach on a grade suitable for haulage purposes. The cost of this preparatory work is obviously a charge against the cost of mining the ore, each ton of which must bear its proportionate share. As the stripping-ore ratio (volume or tons of overburden and capping to tons of ore made available for open-pit mining) increases, obviously the cost of stripping per ton of ore increases.

The question of whether to mine by open-pit or underground methods is an economic one. The economic limit for open-pit mining is reached when the preparatory and stripping costs per ton of ore plus the open-pit mining cost per ton of ore plus the cost of equipment, interest, and amortization charges equal the cost per ton by mining underground plus equipment and development cost and interest and amortization charges.

The stripping-ore ratio is a basic factor used for determining whether to employ open-pit or underground methods and for determining the economic limits of open-pit mining.

Other factors influencing the choice between open-pit and underground methods may be inferred from the advantages and disadvantages of open-pit mining, some of which are listed below. In specific instances consideration of these may affect the choice of method or result in revision of pit limits established by cost estimates based upon stripping-ore ratios.

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[74] Malozemoff, P., Jigging Applied to Gold Dredging: Eng. and Min. Jour., vol. 138, No. 9, September 1937, pp. 34–37.

Among the advantages of open-pit mining are its flexibility, the ability to obtain mass production with it, and the ease with which rate of production can be increased or decreased once the pit has been developed fully; small shut-down expense; the ability to mine selectively to meet requirements for certain grades of ore (except in gloryhole or milling pits); complete extraction of the ore inside the pit limits (except as limited by ore benches for haulage tracks and maintenance of safe pit slopes, but this ore can usually be recovered finally by employing special clean-up methods); comparatively small number of men employed, a large proportion of whom are skilled labor in mechanized operations; and elimination of hazards inherent in underground mining operations.

On the other hand, certain disadvantages may outweigh the advantages and affect direct economic considerations in some instances. Thus, large open-pit operations involve heavy capital outlay for equipment, and where the amount of overburden to be removed is extensive, correspondingly high capital expenditure is required for stripping. This capital is nonproductive until ore mining is begun, and during the stripping period heavy interest charges often accumulate. The time elapsing before production begins may in itself be a serious disadvantage, especially if exploitation is undertaken at a time when ore prices are favorable and the demand for ore is strong. Disposal of the waste from stripping operations sometimes occasions a serious problem, especially when the terrane is flat or where exorbitant prices must be paid for dump area near the mine. Climatic conditions may limit the operation to being seasonal and necessitate cessation of operations during certain months, and in areas where torrential rains are prevalent frequent flooding of the pit may be a serious obstacle to pit mining.

Simple forms of open-pit ore mining include excavation and loading of overburden and ore by hand into wagons or trucks and excavation and removal of overburden and ore by means of animal-drawn slip or drag scrapers. Such methods are applicable to deposits that are too small to warrant capital investment in mechanical equipment and will not be discussed.

Methods in which "bulldozers" are employed for removal of soil and overburden supplemented by the use of power-operated drag scrapers for excavating (with or without previous preparation by drilling and blasting) are applicable to somewhat larger deposits and will receive some further mention.

Overhead slackline cableways are used extensively in quarries, for sand and gravel excavation, and for other purposes but are not used very extensively for ore mining in this country. This kind of work has been discussed by Thoenen [75] in relation to sand and gravel excavation in an earlier Bureau of Mines publication. Discussion in this bulletin is limited to two principal forms of open-pit mining—gloryhole or milling and power-shovel and dragline mining.

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[75] Thoenen, J. R., Sand and Gravel Excavation. Part 2. Power Scraper, Slackline Cableway Excavator, and Hydraulic Giant: Bureau of Mines Inf. Circ. 6814, 1934, 95 pp.

GLORY-HOLE OR "MILLING-PIT" MINING

In glory-hole mining the ore is broken down around one or more raises or mill holes extending upward from an underground haulageway driven below the ore or beneath the ultimate bottom of the glory hole or pit. The pit is thus in the form of an inverted cone or a series of cones the rims of which finally intersect, and the ultimate rim of the pit will conform in shape to the outlines of the ore body or of that portion of the ore body that is to be mined by this method. The sides of the pit sloping toward the mill hole or raise often are mined in narrow benches, the broken ore being blasted into the pit, whence it is drawn off through the mill holes into cars and trammed or hoisted to the surface. When the slopes of the pit become too flat for the ore to run into the mill holes it becomes necessary to use drag scrapers or other means to transfer it.

The glory-hole method has been used extensively for mining relatively small pipe- or wedge-shaped ore bodies or the upper parts of bodies that apex at or near the surface, and for mining the ore remaining in track benches and in the bottom of power-shovel pits below the elevation where locomotive haulage becomes uneconomical because of adverse track grades. Some stripping of loose overburden or barren cap rock may have to be done before glory-hole mining can be started, possibly by the use of scrapers, dragline excavators, or power shovels, by hydraulicking, or by glory-holing.

The cost of equipment for glory-hole mining usually is much less than for power-shovel work, and for relatively small-scale operations the production cost may be less. For large-scale operations, mining by power shovel usually is cheaper. Glory-hole work is not well-adapted to selective mining as all material within the pit must be blasted down into the mills. Thus, lean ore or waste inclusions within the ore can be taken out and disposed of separately only if the ore is first cleaned from the slopes and bottom of the pit and the mill holes emptied, and then the pit is cleaned and mill holes are emptied again after the waste is blasted out.

At the Britannia mine [76] the glory-hole method was employed in the Bluff ore body (fig. 132), where it was large enough to warrant the use of this method. Glory holes were opened, likewise, in the upper Fairview ore body, where closely spaced parallel veins with some mineralization between them made this method preferable to more expensive selective mining methods. Working benches were 3 feet wide and varied in length. The ore was drawn through grizzly chambers into transfer raises leading to the underground transportation system. In the East Bluff ore body low-grade ore around the glory holes and old stopes was mined by shooting the ore out into the pit, using large charges of explosive placed in coyote drifts driven toward the side of the pit from underground workings.

When the open pit at Sacramento Hill, Bisbee, Ariz.,[77] became so deep that hauling ore from the pit was too expensive, the deep ore was mined by glory-holing. In 1929, 574,000 tons and in 1930, 304,000 tons were mined by this method. The bottom of the old pit was oval, about 450 feet long and 400 feet wide. A haulageway 2,000 feet in length and 239 feet below the bottom of the pit was driven from the Sacramento shaft to form a loop under the pit to serve six mill holes, and later five more mills were developed. A grizzly level was driven 135 feet below the pit to connect the mill-hole raises. Grizzly stations or bulldozing chambers were 8 feet wide, 7 feet high, and 15 feet long, and grizzly bars were of 100-pound rails spaced 17 inches apart. Three-compartment L-shaped raises were driven from the grizzly chambers to a height of 90 feet, or within 35 feet of the bottom of the pit. Holes were drilled from the pit over each raise with a churn drill and blasted. Six-inch churn-drill holes, 60 feet deep and 18 to 23 feet apart around the mills, were blasted to funnel the tops of the mills, forming craters, around the slopes of which holes were drilled 18 to 25 feet deep with tripod-mounted pneumatic drills.

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[76] Brennan, C. V., the Mining Operation at the Property of the Britannia Mining and Smelting Co., Ltd., Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6815, 1934, p. 19.

[77] Gardner, E. D., Johnson, C. H., and Butler, B. S., Copper Mining in North America: Bureau of Mines Bull. 405, 1938, pp. 239-241.

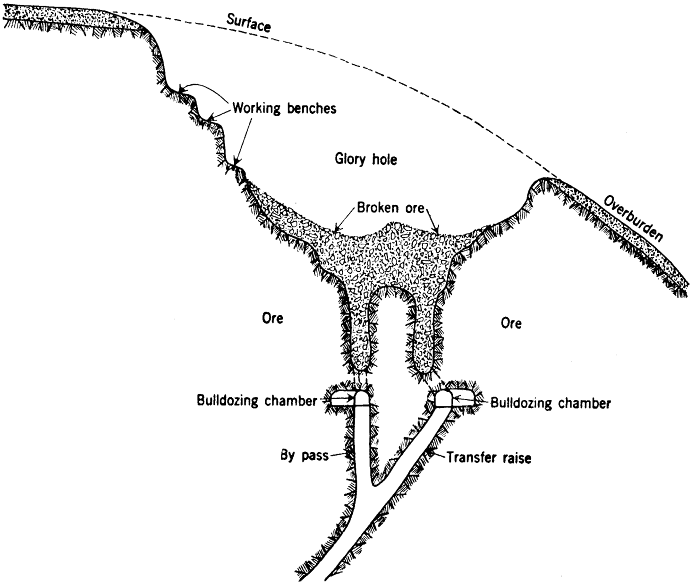

FIGURE 132.—Vertical section of glory hole, Britannia mine, British Columbia.

Figure 133 is a typical section of a glory hole in the Hilltop ore body at Fresnillo, Zacatecas, Mexico.[78] The first cut in each cycle was taken at the bottom of the crater around the raise, and successive benches 8 to 10 feet high were then mined; the direction of retreat was upward toward the rim. The safe limit of glory-hole slope varies greatly in different localities, but at Fresnillo slopes steeper than 70° were carried safely where they were not more than 100 feet high. Drilling was done with light jackhammers, the holes averaging 9 feet in depth. About 120 feet of hole was drilled per machine shift at a contract price of $0.028 to $0.036, United States currency. The average production per machine shift was 130 tons. The waste to ore ratio was 0.017 to 1.

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[78] Livingston, A., Mining Methods and Costs at Fresnillo, Zacatecas, Mexico: Bureau of Mines Inf. Circ. 6661, 1932, 31 pp.

Figure 134, *A*, is a cross section of the New Idria (Calif.) ore body, and figure 134, *B*, is a plan of the 100-foot level of the pit.[79] The stripping was done by glory holing and the ore was then mined by power shovels. The open-cut method was adopted to mine that portion of the ore body between the 300-foot level and surface. Early in 1931 stripping of the hanging-wall waste by glory holing had been in progress for 3 years, during which 1,500,000 tons of rock had been stripped and transferred to the dump by a 42-inch conveyor belt. About 1,500,000 tons remained to be stripped to permit mining to the 300-foot level. To prepare for stripping, a raise was driven in the hanging wall from the 300-foot level to the surface. At a point 27 feet above the level a bulldozing chamber was excavated and grizzlies installed. From the grizzly the rock was fed to the conveyor belt and transported to the dump; 1,000 tons per hour can be handled, but the normal feeding rate was 350 tons per hour. Distance from center of head pulley to center of tail pulley was 1,250 feet; 670 feet of the belt was in the adit, and the rest extended over the dump.

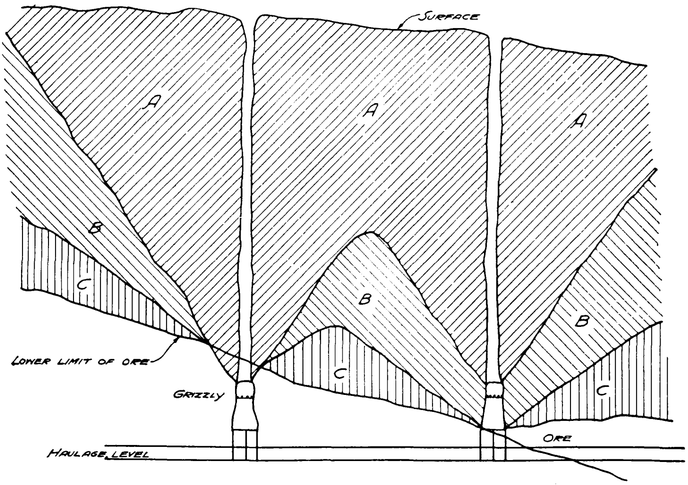

FIGURE 133.—Typical section of glory hole in Hilltop ore body, Fresnillo, Mexico. *A*, First stage. *B*, Second stage. *C*, Final stage.

In the earlier years of open-pit mining on the Mesabi range in Minnesota, a number of milling pits were in operation for mining below levels too deep to be reached by locomotive haulage.

Table 53 presents data on seven typical glory-hole operations.

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[79] Moorehead, W. R., Methods and Costs of Mining Quicksilver at the New Idria Mine, San Benito County, Calif.: Bureau of Mines Inf. Circ. 6462, 1931, 14 pp.

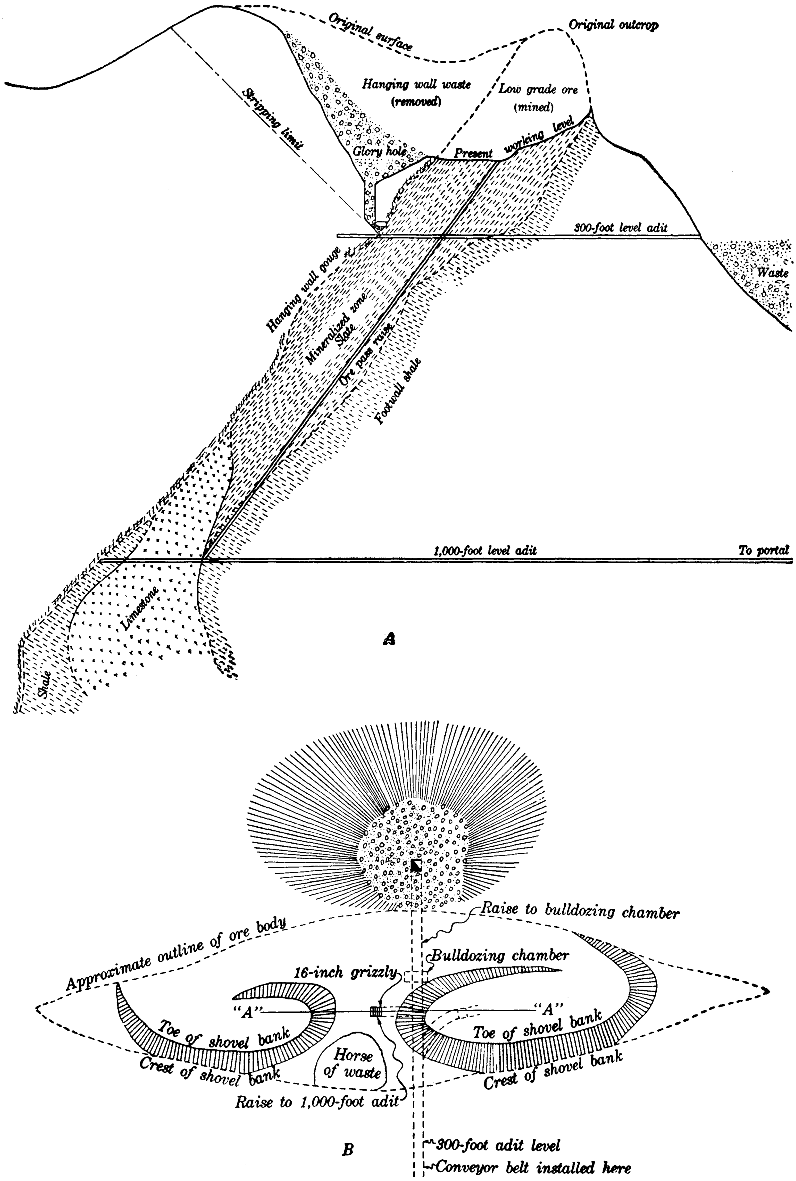
Original surface
Original outcrop
Hanging wall waste (removed)
Low grade ore (mined)
Stripping limit
Glory hole
Present working level
300-foot level adit
Waste
Hanging wall gouge
Mineralized zone
Slate
Ore pass raise
Footwall shale
1,000-foot level adit
To portal
Limestone
Shale
A
Raise to bulldozing chamber
Approximate outline of ore body
Bulldozing chamber
16-inch grizzly
"A"
"A"
Toe of shovel bank
Toe of shovel bank
Horse of waste
Crest of shovel bank
Crest of shovel bank
Raise to 1,000-foot adit
B
300-foot adit level
Conveyor belt installed here

FIGURE 134.—*A*, Cross section of New Idria (Calif.) ore body. *B*, Plan of New Idria pit, 100-foot level.

TABLE 53.—*Data on typical glory-hole operations*

Mine and year	Nature of ore or waste	Approximate rate of production, tons per month	Direct glory-hole cost per ton							Tons per machine shift	Tons per man-shift
			Labor	Compressed air drills and steel	Power	Explosives	Other supplies	Haulage	Total		
Cananea, Mexico, 1931	Hard porphyry (waste for stope filling).	9,000	$0.073	$0.012		$0.056	$0.003	$0.012	$0.156		45.40
Hilltop, Fresnillo, Mexico, 1921–25	Shale and graywacke; easy drilling and breaking.	80,000	.136		$0.029	.057	.043		.265	130	12.76
Catillas & San Carlos, Fresnillo, Mexico, 1921–25.	Shale and graywacke (broken ore, sticky).	92,000 ore; 27,000 waste	.148		.029	.058	.028		.263		12.00
New Idria, Calif., Sept. 1930	Waste, soft shale, and serpentine	65,000	.070			.039	.005		.114		
San Carlos, Mexico	do	3,300	.055								
Tintic Standard, Utah	Waste for stope filling		.200								
Copper Queen (Sacramento), Ariz., 1929–31.	Porphyry ore	48,000 (1929); 25,000 (1930)	.089	[1].028		.052	.022		[2].191		
Britannia, Fairview, British Columbia.[3]	Sulfide ore in sheared volcanics	105,000	.120	.003	.018	.008	.012	.077	[4].238		

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[1] Drills and tools only.

[2] Exploration and development, tramming, hoisting, drainage, sanitation and ventilation, sampling and assaying, and mine department expense, $0.326; proportion of general department expense and haulage to concentrator, $0.091: total production and transportation cost, $0.608.

[3] Data cover ore mined by direct glory holing and by coyote blasting (see text).

[4] Additional costs were development, $0.008; general underground and surface expense, $0.070; rail haulage to mill, $0.054; making total production cost, $0.370. Costs cover combined glory-hole and coyote drift methods.

POWER-SHOVEL AND DRAGLINE PITS

The determination of mining and stripping limits prior to undertaking exploitation by power-shovel methods requires careful engineering and often involves complicated computations.

ENGINEERING ESTIMATES

Stripping of overburden and capping in open-pit mining corresponds to development work in underground mining and, as previously pointed out, is a part of the ore-production cost. The cost of stripping is therefore an important factor in the calculations. The unit cost per cubic yard or per ton of stripping material to be removed will vary between wide limits, depending on the depth and character of the capping and overburden, topography, total volume and rate at which it is to be stripped, distance to the spoil dump and grade of track or truck roads, method of transportation, size, capacity, and type of excavating and haulage equipment, wage rates, and local costs of supplies and power.

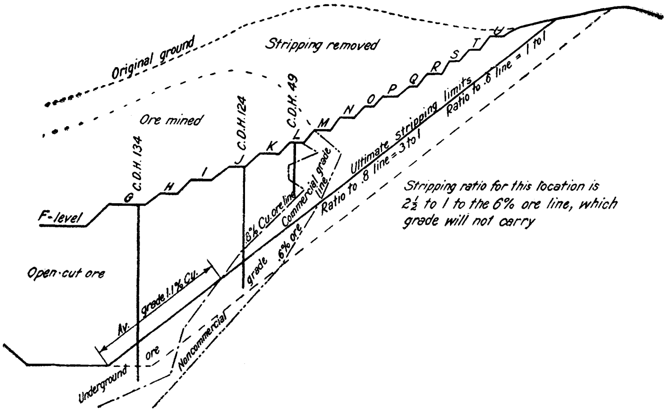
Original ground
Stripping removed
Ore mined
C.D.H. 134
C.D.H. 124
C.D.H. 49
F-level
G
H
I
J
K
L
M
N
O
P
Q
R
S
T
U
Ultimate stripping limits
Ratio to .6 line = 1 to 1
Ratio to .8 line = 3 to 1
Stripping ratio for this location is 2½ to 1 to the 6% ore line, which grade will not carry
.8% Cu. ore line
Commercial grade line
Open-cut ore
Av. grade 1.1% Cu.
grade .6% ore
Noncommercial
Underground ore

FIGURE 135.—Typical section showing method of fixing the ultimate stripping limits, Bingham, Utah.

The total volume of stripping will be that vertically above the limits of the ore pit plus that outside of the ore pit necessary to maintain safe strip-pit slopes and benches and provide working approaches to the pit. The strip-pit slopes will be governed by the character and depth of the capping or overburden and the direction of major planes of weakness, such as faults, bedding planes, and schistosity. The volume of stripping outside the ore-pit area and, hence, the stripping-ore ratio will increase rapidly with the depth of capping or overburden, especially if the surface rises away from the pit (see fig. 135). Since the stripping-ore ratio determines the stripping charge per ton of ore, it limits the grade of ore that can be mined profitably by open-pit methods and hence determines the boundary of the ore pit. Metal prices, which determine the grade of ore that can be mined profitably, are subject to wide fluctuations. Hence, it is evident that a careful appraisal of future market trends is as essential as precision in engineering calculations. Balancing of the various factors therefore often requires the use of cut-and-try methods.

Soderberg [80] has discussed the estimates involved at the Utah copper pit, and the following is abstracted from his paper as a good example of the factors that must be considered and the methods to be used in making open-pit estimates.

Preparatory to making detailed ore estimates, the cut-off between commercial ore and waste must be determined; that is, a grade must be determined below which the material cannot be mined and meet its mining and treatment costs and show a profit. To arrive at this cut-off grade certain assumptions must necessarily be made, such as the selling price of copper, the estimated recovery in percent of gross content, and the cost of producing a pound of copper, which includes all costs other than stripping. The stripping cost is kept separate for reasons that will develop later. Table 54 is set up to illustrate the method used.

TABLE 54.—*Method of determining point of cut-off between ore and waste*

1 Copper, percent	2 Gross pounds per ton	3 Assumed recovery, percent	4 Pounds recovered	5 Assumed selling price per pound, cents	6 Assumed cost per pound less stripping, cents [1]	7 Partial net profit per pound, cents	8 Loss per pound, cents
1.10	22	91	20.0	13.5	6.25	7.25	
1.00	20	90	18.0	13.5	6.94	6.56	
.90	18	89	16.0	13.5	7.81	5.69	
.80	16	87	13.9	13.5	8.99	4.51	
.70	14	85	11.9	13.5	10.50	3.00	
.60	12	83	10.0	13.5	12.50	1.00	
.50	10	80	8.0	13.5	15.62		2.12

1 Copper, percent	9 Stripping cost per cubic yard, cents	10 Ratio of waste to ore [2]	11 Proportionate stripping cost per pound this basis, cents	12 (6+11) Total cost per pound, cents	13 Profit per pound, cents	14 Loss per pound, cents
1.10	40	½-1	0.50	6.75	6.75	
1.00	40	½-1	.56	7.50	6.00	
.90	40	½-1	.62	8.43	5.07	
.80	40	½-1	.72	9.71	3.79	
.70	40	½-1	.84	11.34	2.16	
.60	40	½-1	1.00	13.50		
.50	40	½-1	1.25	16.87		3.37

[1] Based on a cost of $1.25 per ton. These figures can be varied to cover increasing costs due to increasing copper content, such as bullion, freight, refining, selling, etc.

[2] Or ¼ yard waste to 1 ton of ore.

Under this set of conditions a copper content of 0.6 percent would be the point of cut-off; any grade under this would be waste and anything over should be classed as ore. Other tables should be made using other figures for doubtful assumptions to assist in establishing a safe cut-off figure.

At this stage it is convenient to set up a table of grades showing the amount of stripping any given grade of ore will carry. This is usually done as follows:

If it costs 40 cents to waste a cubic yard of overburden weighing 2 tons, ½ ton will cost 10 cents. It is then necessary to determine the grade of ore that will yield a return of 10 cents under the given conditions, assuming the average recovery to be 85 percent.

10 cents divided by 13.5 cents =0.741
0.741 divided by .85 = .872
.872 divided by 2,000 (pounds per ton) = .0436 percent copper.

By adding this increment of grade (0.044) to the ore, it will support the removal of an additional half ton of waste for each addition, from which the following table may be set up:

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[80] Soderberg, A., Mining Methods and Costs of the Utah Copper Co., Bingham Canyon, Utah: Bureau of Mines Inf. Circ. 6234, 1930, pp. 6-10.

Tonnage of stripping, plus all other costs, carried by 1 ton of ore of various grades

Grade of ore, percent	*Tonnage of stripping*
0.6	0.5
.644	1.0
.688	1.5
.732	2.0
.776	2.5
.820	3.0

From tables 53 and 54 graphs may be drawn from which it can be determined at a glance whether or not a certain block of material is ore or waste when the stripping ratio has been determined.

Dividends, of course, cannot be paid on a grade of ore at or near the point of cut-off; the average grade, therefore, must be well in excess of the cut-off grade, and no section of the ore body that will not pay its own way should be combined with higher grades for the purpose of increasing the reserve. Possible exceptions to this rule appear, of course, when a "horse" of waste or a small amount of low-grade material occurs that has to be removed in any case. These small quantities of waste may not be easily separated and may be milled at a loss (capacity permitting), which, however, will be smaller than the cost of removing the material as waste. In such cases, the low-grade tonnage is included in the ore reserve with its grade. The engineer's judgment will guide him (after he has made a complete analysis of the ore body) in rounding out an estimate where so many variables are concerned. It is well to remember that material which at the time of the estimate is waste may come into the classification of ore by an increase in the price of copper, by an improvement in metallurgy, or by a lowering of costs with improved equipment.

A point that is never lost sight of is that the total cost of mining and stripping shall not exceed a reasonable underground cost. To determine this cost limit a series of trial sections was made up running normal to a tentative location of the stripping limits. On these sections detailed studies were made showing the ratio of ore to waste and the stripping limit for the particular section determined (see fig. 135 for a typical example of the problems involved). In this study it was necessary to determine the grade of the ore in each section and to ascertain just what amount of stripping could be moved and still show a commercial profit from the ore in question. Referring to the table of stripping ratios, it may be noted that under the costs and conditions upon which the table is based, ore having a grade of 0.82 percent copper can carry a stripping ratio of 3 to 1 or of 1½ cubic yards of waste to each ton of ore. At 40 cents per cubic yard this means a stripping cost of 60 cents per ton of ore. In some sections the average grade of ore is in excess of 1 percent copper and as far as grade is concerned could still show a profit for larger stripping ratios permitting the moving of the stripping limit to the increased ratio. But stripping costs in excess of 60 cents per ton under the conditions mentioned, together with other mining costs, exceed the probable underground mining cost; therefore, the maximum stripping ratio should not be greater than 3 to 1 and ore outside such limits should be classed as underground ore. Referring again to the section, it may be noted that the 0.6-percent ore line extends beyond the stripping limit, but such a grade is not considered profitable by underground mining and is therefore not included in ore-reserve calculations. When located within stripping limits, this grade can be classed as commercial ore and included in the reserve.

The study was continued for each section; the ultimate location of stripping limits was then finally laid out, and it was found that of the 625,000,000 tons of ore reserve developed to date, approximately 580,000,000 could be removed by open-cut methods. Possibly some 40 years hence, after open-cut methods have ceased to be profitable, there will be a "mop-up" job to win the remaining tonnage by caving methods. This will entail considerable development work, and plans that are being made for shoveling below the present scene of operations are being laid out to tie in with the possible underground operations.

In addition to the above factors that determine the choice between open-cut and underground methods is the practical side involving the necessity for mass production of a low-grade ore to make it of maximum commercial value. To this must be added the value of flexibility of control of production. To illustrate: If occasion should demand an immediate increase in production to 60,000 tons from a mine ordinarily producing 50,000 tons of ore per day, practically all that is involved in open-cut work is to take two shovels working on stripping and place them on ore. In underground work, to increase the number of ore faces 10 percent would present a serious problem.

Other issues involved that fix the limit to which open-cut operations can be carried are such factors as maximum degree of railroad curvatures, sufficient space for the efficient operation of power shovels, adjacent property rights, dump grounds for waste material, and above all the safe degree of over-all slope. Calculations of ore tonnages recoverable by open-cut methods and stripping are also dependent upon the slope.

The total ore removed to January 1, 1929, amounts to 175,007,974 tons having an average grade of 1.21 percent copper. During the same period 94,338,953 cubic yards of capping and low-grade material has been stripped and dumped in nearby gulches. This gives a stripping ratio to date of 1.1 tons of waste to 1 ton of ore, and is also the ratio being maintained at the present time. The final ratio is entirely dependent upon the ultimate over-all pit slope. Based upon a 40° slope this ratio will be approximately ½ to 1, but, if conditions make it necessary to use a much flatter slope, the ratio may be increased to equal amounts of waste and ore. These are general averages, as there are sections where the ratio reaches the maximum of 3 to 1, and it is in such places that slope is of paramount importance. When it is considered that in one section of the Utah Copper pit the question arose as to whether or not 38° instead of 40° should be used above a certain level and that the cost of removing this extra amount of waste would reach a sum of $2,000,000, it can readily be seen how vital a thing, at least to the Utah Copper Co., the ultimate over-all slope really is.

The individual bench slope—that is, the actual slope of one shovel face from the upper edge to the toe of the slope—is of less consideration. This is, of course, a function of the over-all slope, but while an individual bench might stand at 60° from the horizontal, it is not to be concluded that a face 1,500 feet high will stand at the same angle of repose. When the material in any part of the ore body gradually declines in grade from ore to waste, there is always the question as to where the limit of excavation will ultimately be, and it therefore becomes necessary to maintain the shovel terraces on the pit face where, for the time being at least, operations are suspended. This shovel bench can be maintained at a minimum width to accommodate a shovel and loading track—say 30 feet. The slope we are most concerned with is the aggregate made up of the individual bench slopes plus the width of benches. In other words, the over-all slopes would be the angle from the horizontal from the top edge of the excavation to the bottom toe of the excavation, and it is this angle that the Utah Copper Co. has tentatively set at 40°.

There always will be local variations. At some points the slopes will doubtless take the angle of repose of broken material, say 35°. With others a slope of 50° may be safe. Geological conditions will enter into this. For instance, where stripping is being done against a face of quartzite that is dipping toward the operations at 30° from the horizontal, this slope will doubtless turn out to be the dip of the beds. On the contrary, where the beds are dipping into the bench, say from the shovels, a much steeper slope can be maintained.

EVOLUTION OF EQUIPMENT AND PRACTICES

Since late in the last century, power shovels and draglines for digging and loading and locomotives and cars for hauling from the pit have been in general use at most large-scale open-pit operations. Starting with small steam shovels and with dump cars drawn by teams or "dinky" locomotives over narrow-gage tracks, the size and capacity of the equipment was increased, design and efficiency of equipment and methods and equipment for drilling and blasting were improved, and advances were made in other practices as operations were undertaken on a larger and larger scale and as the depths of stripping and ore pits became greater.

Typical of the evolution in equipment and methods are the developments on the Mesabi range, as described by Hunner.[81] The following notes are abstracted from Hunner's paper:

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[81] Hunner, Earl E., Some Recent Developments in Open-Pit Mining on the Mesabi Range : Trans. Am. Inst. Min. and Met. Eng., Yearbook, 1930, pp. 106–128.

In 1903 one contractor at Chisholm, Minn., was using 60- and 70-ton railroad-type steam shovels with 2½- and 3-cubic-yard dippers for loading stripping into narrow-gage, 4-cubic-yard wooden dump cars hauled by saddleback, 8- to 18-ton steam locomotives; ore was loaded into railroad cars hauled by 45- to 70-ton, six-wheel, switching-type locomotives.

In 1914 a contract was let for removing 8,000,000 cubic yards of stripping at the rate of 2,000,000 cubic yards per year to uncover 6,000,000 tons of ore for open-pit mining. It was contemplated that later work would require the additional removal of 14,000,000 cubic yards of overburden to make available for mining an additional 19,000,000 tons of ore. In 1917 the contractors on this job were using the following equipment, typical of the best practice at that time:

9 six-wheel Baldwin steam switching locomotives, 20 by 26 inches.
8 Kilbourne and Jacobs 20-cubic-yard automatic air-dump cars.
49 Kilbourne and Jacobs 16-cubic-yard automatic air-dump cars.
1 eight-wheel, 25-ton locomotive crane.
1 model 100 Marion steam shovel, 5-cubic-yard dipper, weight 137 tons.
2 model 76 Marion steam shovels, 4-cubic-yard dipper, weight 107 tons.
1 model 36 Marion revolving shovel, 1½-cubic-yard dipper, weight 40 tons.

By 1919 one 300-ton revolving steam shovel with 6- and 8-yard dippers had been added to the equipment, which in one month of two-shift operation removed 250,000 cubic yards of sand and gravel overburden. Tracks were laid with 80-pound rails on 1.5 to 2.5 percent main-line grades.

Large revolving shovels were found to have decided advantages over 100- to 110-ton railroad-type shovels under certain conditions. In 1919 the M. A. Hanna Co. purchased two electric, full-revolving shovels (one Marion 300-ton and one Bucyrus 225–B) each equipped with 150-foot boom and complete dragline equipment. They used 8-cubic-yard buckets for stripping and 6-cubic-yard buckets for ore when equipped as shovels. As draglines they used 5-cubic-yard Page buckets. Power for shovel operation was supplied at 22,000 volts and stepped down to 2,200 volts.

In 1924 a Bucyrus 50–B revolving electric shovel mounted on caterpillar crawlers and equipped with a ¾-yard bucket was installed at the Wabigon mine and as Bucyrus 80–B revolving, electric, caterpillar-traction shovel at the Richmond mine, Palmer, Mich.

In 1926, two Bucyrus 120–B revolving, electric, caterpillar-traction shovels with 4-cubic-yard buckets were placed in operation at the Susquehanna mine, Hibbing, Minn. These three shovels were the first of their type and size using direct-current motors to be employed in the Lake Superior district. Since then shovels of this type have come into general use on the Mesabi range, most of them being equipped with 4- to 5-cubic-yard dippers.

In 1924 the first installation of standard-gage electric locomotives in open-pit ore mining in this country was made at the Wabigon mine, Buhl, Minn. The equipment comprised the following:

Electric shovels.—One Marion 300–E and one Bucyrus model 50–B, both equipped with direct-current motors operated by motor-generator sets; the model 300–E had full Ward-Leonard control, whereas the 50–B had partly rheostatic control.

Electric locomotives.—Three 60-ton General Electric swivel-truck type designed to operate at 600 volts direct current with capacity to haul a train of 340 tons (including locomotive) on a tangent track against a 3-percent adverse grade at 7½ miles per hour. Each locomotive is equipped with an air-operated pantagraph sliding-bow collector mounted on the cab for use on main tracks and with two air-operated side-arm collectors, one on each side, for use with side trolley wire on loading and dump tracks.

Trolley-wire system.—An overhead trolley-wire system was adopted on account of first cost, freedom from obstruction in the event of derailments and spill of stripping along the tracks, and greater flexibility and safety than a third-rail system.

In 1928–29 a similar installation was made at the Mesabi Chief mine, Keewatin, Minn., using one Bucyrus 225–B 90-foot-boom, 8-cubic-yard-dipper, full-revolving shovel, and two Bucyrus 120–B 32-foot-boom, 4-cubic-yard shovels, all full-revolving type with caterpillar traction and equipped with direct-current motors and Ward-Leonard control. General Electric 60-ton 8-wheel locomotives were used. The rolling stock comprises 16 air-dump side-pivot, drop-door cars of 30 cubic yards capacity. A Bucyrus heavy-type spreader is employed on the dumps for spreading the spoil and a Nordberg track shifter for throwing tracks and for general utility work.

Although the foregoing summarizes briefly the advances made up to about 1930 by only one company, the changes are typical and illustrate the evolution in modern excavation and haulage equipment for open-pit metal mines.

Electric pit haulage has not been adopted generally in the Lake Superior district chiefly because there is a large amount of serviceable and efficient steam equipment on hand, and the remaining life of the pits is too short to justify the capital outlay for electrification. The revolving caterpillar-traction electric shovel, however, has become standard because the savings in operating cost through its use are so great that it has paid to scrap steam-operated shovels. Among its advantages are its mobility without use of tracks, eliminating the pit crew required with the old railroad-type shovel (the shovel crew consists usually of only 2 men); elimination of delays to loading when coaling; ability to dig and load in any direction; superior efficiency, control, and power characteristics of electric shovels.

Track shifting and grading, spreading of spoil on the stripping dumps, and clean-up of stripping on top of the ore were formerly done by hand; this work is now done principally by locomotive cranes, bulldozers, and dirt spreaders.

Bank drilling, formerly done by hand with "jump drills," long-handled shovels, or post-hole diggers in soft ore, or by piston drills, is now done principally with mobile, caterpillar-mounted, electric churn drills from the top of the bank along the berms or benches of the pit.

More recently, motorized pit haulage (gasoline- or Diesel-powered trucks or trucks and trailers) has been adopted for the smaller pits and for clean-up work in larger pits where adverse grades are too severe for locomotive haulage. In other districts large operations have been equipped at the outset for motorized haulage, large trucks of 30 to 35 tons capacity being used. Truck haulage has the advantage of flexibility, elimination of tracks and track laying, and ability to negotiate steep grades and to load in restricted space in corners and local dips in the pit. For long, main-line hauls in the pit and from the pit to beneficiation plant, it is probable that locomotive haulage is generally more economical and combinations of truck and rail haulage (dual haulage system) may prove most satisfactory under some conditions, using truck haulage to bins on the main line or to the crest of the pit, followed by rail haulage to the plant. Based upon a recent investigation of truck vs. rail haulage in bituminous-coal mines, Toenges and Jones [82] conclude that with equipment in its present stage of development and within the range of conditions studied, for a new mine a dual haulage system would effect a saving in transportation cost if the round-trip haul exceeds 2.7 miles.

The first use of trucks for large-scale open-pit haulage in metal mines in the United States was, so far as the authors are aware, at the United Verde mine. According to Alenius,[83] the original plan was to mine the material below the 160-foot level by glory-holing, and in 1924 several glory holes were started. They did not prove practical for the following reasons:

1. Inability to drive raises in the desired locations on account of the underground fire and existing stopes.
2. Dilution of ore and inability to sort material properly.
3. Indefinite production.
4. Operating hazards.

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[82] Toenges, Albert L., and Jones, Frank A., Truck vs. Rail Haulage in Bituminous-Coal Strip Mines: Bureau of Mines Rept. of Investigations 3416, 1938, 54 pp.

[83] Alenius, E. M. J., Methods and Costs of Stripping and Mining at the United Verde Open-Pit Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6248, 1930, 34 pp.

Early in 1925 a plan was devised by W. V. De Camp, then assistant general manager, by which the lower pit would be mined by small shovels working on low benches. The material was loaded into trucks, which dumped into several transfer raises extending from the 1,000-foot mine level. Figure 136 shows a plan of the pit as of July 1, 1929, and figure 137 is a typical section through the No. 3 raise showing dumps on three levels. The major equipment employed was as follows:

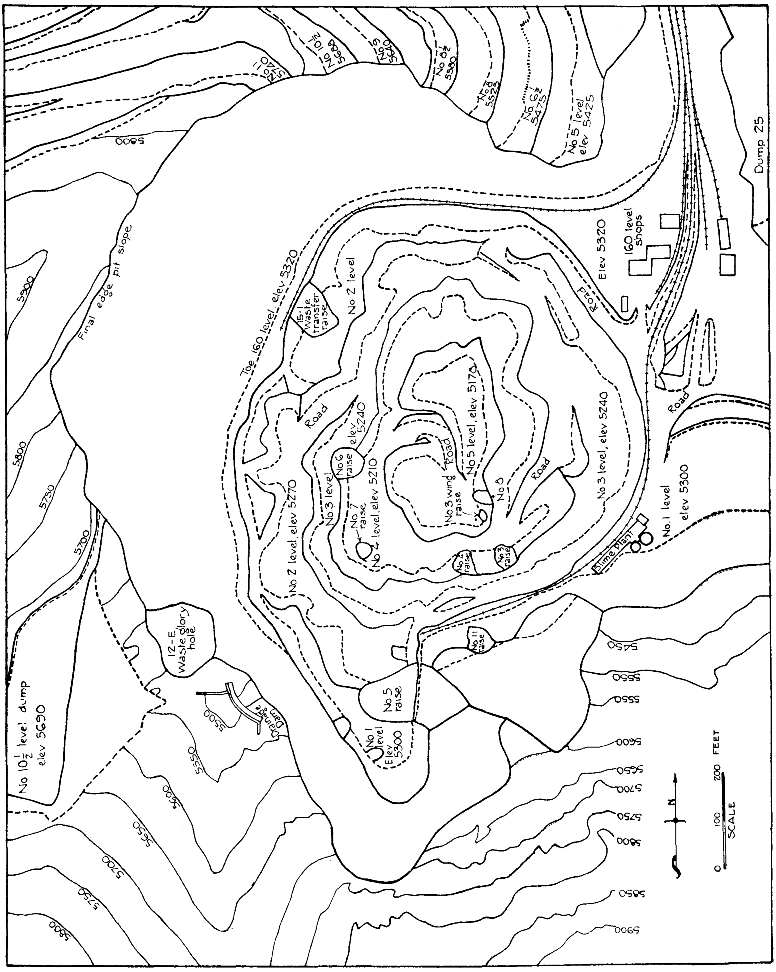

No 6½ 5580
No 7 5525
No 6½ 5475
No 5 level elev 5425
Dump 25
160 level shops
Elev 5320
Road
Toe 160 level, elev 5320
15-1 Waste transfer raise
No 2 level
Final edge pit slope
5900
5800
5750
5700
Road
No 6 raise
elev 5240
No 5 level, elev 5178
Road
No 3 wing raise
No 8
Road
No 3 level, elev 5240
No 1 level elev 5300
No 3 level
No 7 raise
No 4 level, elev 5210
No 2 level, elev 5270
No 2 raise
No 3 raise
Slime plant
12-E Waste glory hole
No 11 raise
No 5 raise
No 1 level Elev 5300
No 10½ level dump elev 5690
Drainage Dam
5450
5500
5550
5600
5650
5700
5750
5800
5850
5900
N
0 100 200 FEET
SCALE

FIGURE 136.—Plan of United Verde pit, Arizona, July 1, 1929.

4 electric, full-revolving, 1¾-cubic-yard shovels.

10 six-wheel, four-wheel rear drive, 4-cylinder, 10-ton trucks and one 6-cylinder, 10-ton truck.

6 tractor trucks with 2 front wheels and caterpillars on rear, 6 cylinder, 10-ton capacity.

2 four-wheel, 4-cylinder, 5- to 6-ton trucks.

1 four-wheel, 6-cylinder, 3½-ton truck.

1 four-wheel, 6-cylinder, 2-ton truck.

1 four-wheel, 6-cylinder, 1-ton truck.

1 gasoline-powered, full-revolving crane.

1 tractor-powered road scraper.

During 1928, 167 cubic yards or 390 tons of material was trammed per truck-shift, the average one-way haul being 500 feet. Additional data on this and other tractor operations are given later.

Recently several open-pit mines have installed conveyor-belt systems for transporting the ore from gathering points in the pit to bins on surface outside the pit limits. Excavation is by power shovel or by tower drag scrapers. When power shovels are used they load into trucks that haul the ore to raises leading to the conveyors, and when drag scrapers are used, the ore is dragged to the raises by hoists. Both systems are employed at the Spruce mine, Eveleth, Minn., where the installation is believed to be the largest of this kind in an American metal mine, although others were planned when it began operation in 1937. The system was described in Skillings' Mining Review, October 16, 1937, pp. 1–2, 10–11, et seq. Figure 138 is a longitudinal section of the system in which the lower part is the continuation to the left of the upper part. The general arrangement is shown in plan in figure 139, figure 140 is a section of a truck-dumping station, and figure 141 is a section of a tower- or drag-scraper station. The ore in each case is delivered to jaw crushers, whence it passes through transfer raises to pan feeders that deliver it to the conveyor belts. There are nine belts having a combined length of 4,481 feet, of which 3,170.5 feet are in underground drifts. All belts are 33 inches wide, and when traveling at 500 feet per minute the rated capacity is 500 tons per hour. General specifications of the equipment are given in figures 139, 140, and 141. Although operating costs are not available to the authors at this time (August 1938), careful estimates by engineers of the company prior to beginning

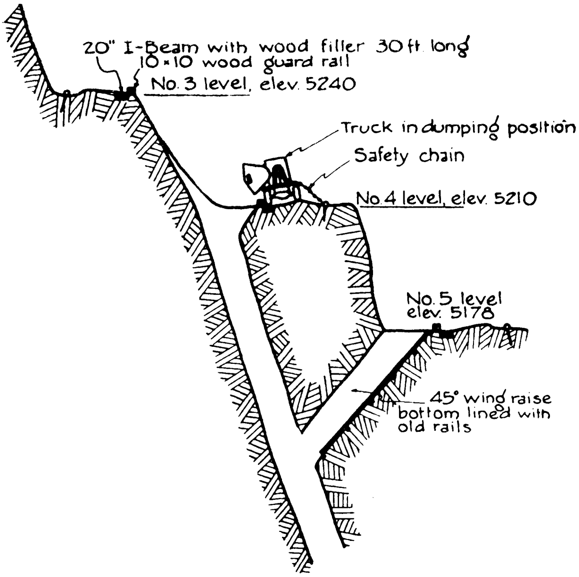

20" I-Beam with wood filler 30 ft long
10×10 wood guard rail
No. 3 level, elev. 5240
Truck in dumping position
Safety chain
No. 4 level, elev. 5210
No. 5 level elev. 5178
45° wind raise bottom lined with old rails

FIGURE 137.—Typical section through No. 3 raise, showing dumps on three levels, United Verde mine, Arizona.

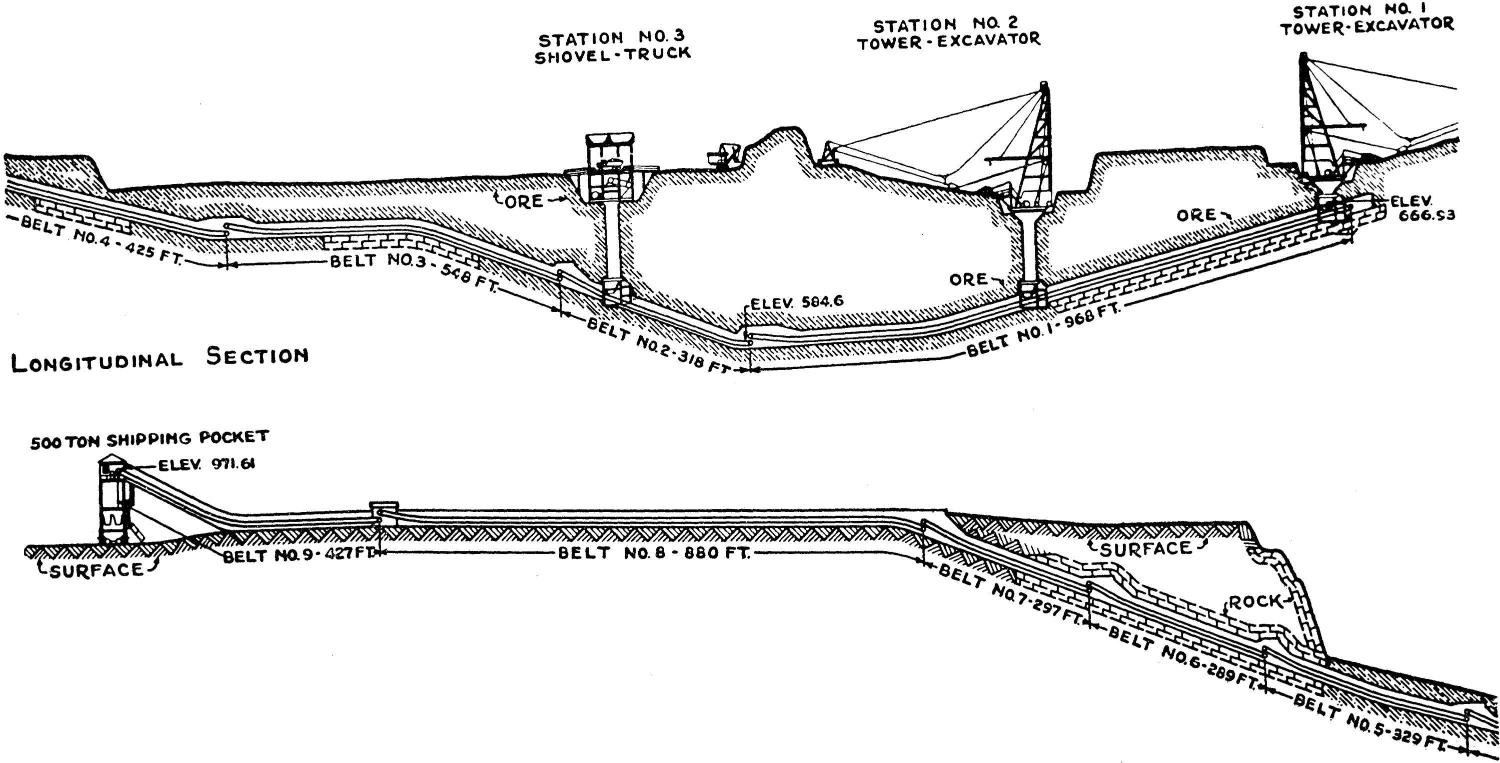
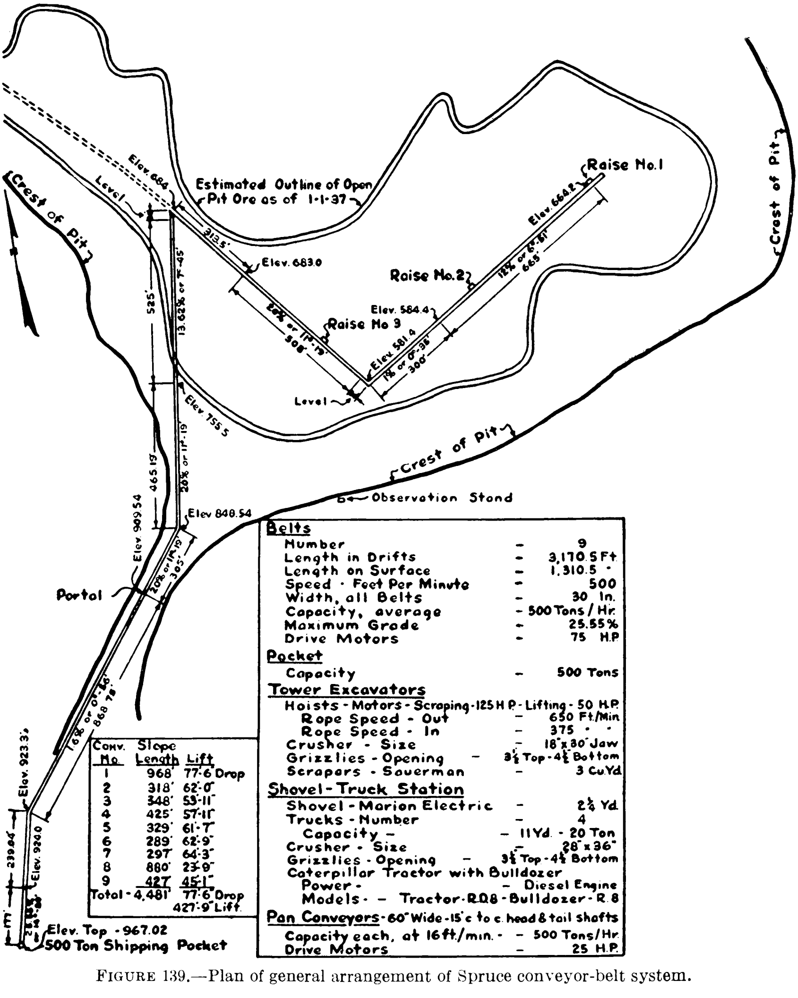

STATION NO. 3
SHOVEL-TRUCK
STATION NO. 2
TOWER-EXCAVATOR
STATION NO. 1
TOWER-EXCAVATOR
ORE
ORE
ORE
ELEV. 666.53
ELEV. 584.6
BELT NO. 4 - 425 FT.
BELT NO. 3 - 548 FT.
BELT NO. 2 - 318 FT.
BELT NO. 1 - 968 FT.
LONGITUDINAL SECTION
500 TON SHIPPING POCKET
ELEV. 971.61
SURFACE
BELT NO. 9 - 427 FT.
BELT NO. 8 - 880 FT.
SURFACE
ROCK
BELT NO. 7 - 297 FT.
BELT NO. 6 - 289 FT.
BELT NO. 5 - 329 FT.

FIGURE 138.—Spruce open-pit belt-conveyor system, Eveleth, Minn.

operations indicated a saving of 9 cents per ton as compared to locomotive haulage.

Hoisting of ore in skips running on inclined tracks on the side of the pit or in a shaft connected to the bottom of the pit has been practiced for many years. This practice is adapted particularly to small pits, where the relative volume of stripping or where track or truck-road grades would be prohibitive for haulage purposes, and to the final stage of power-shovel pit operation when mining out the benches and bottom ore.


Estimated Outline of Open Pit Ore as of 1-1-37
Raise No. 1
Raise No. 2
Raise No 3
Crest of Pit
Crest of Pit
Crest of Pit
Elev. 688
Level
313.5'
Elev. 683.0
20% or 11°-19'
508'
Elev. 664.2
12% or 6°-51'
665'
Elev. 584.4
Elev. 581.4
1% or 0°-36'
300'
Level
525'
13.62% or 7°-45'
Elev. 755.5
465.19
20% or 11°-19'
Elev. 909.54
Elev 848.54
Observation Stand
Portal
20% or 11°-19'
305'
1.6% or 0°-56'
868.78
Elev. 923.3
239.04
Elev. 924.0
Elev. Top - 967.02
500 Ton Shipping Pocket

Conv. No.	Slope Length	Lift
1	968'	77'-6" Drop
2	318'	62'-0"
3	348'	53'-11"
4	425'	57'-11"
5	329'	61'-7"
6	289'	62'-9"
7	297'	64'-3"
8	880'	23'-9"
9	427'	45'-1"
Total -	4,481'	77'-6" Drop 427'-9" Lift

Belts	
Number	9
Length in Drifts	3,170.5 Ft.
Length on Surface	1,310.5 "
Speed - Feet Per Minute	500
Width, all Belts	30 In.
Capacity, average	500 Tons / Hr.
Maximum Grade	25.55%
Drive Motors	75 H.P.
Pocket	
Capacity	500 Tons
Tower Excavators	
Hoists - Motors - Scraping - 125 H.P. - Lifting	50 H.P.
Rope Speed - Out	650 Ft./Min
Rope Speed - In	375 " "
Crusher - Size	18" x 30" Jaw
Grizzlies - Opening	3½" Top - 4½" Bottom
Scrapers - Sauerman	3 Cu. Yd.
Shovel-Truck Station	
Shovel - Marion Electric	2¼ Yd.
Trucks - Number	4
Capacity	11 Yd. - 20 Ton
Crusher - Size	28" x 36"
Grizzlies - Opening	3½" Top - 4½" Bottom
Caterpillar Tractor with Bulldozer	
Power	Diesel Engine
Models	Tractor - RD8 - Bulldozer - R.8
Pan Conveyors - 60" Wide - 15' c to c. head & tail shafts	
Capacity each, at 16 ft./min.	500 Tons/Hr.
Drive Motors	25 H.P.

FIGURE 139.—Plan of general arrangement of Spruce conveyor-belt system.

DATA ON POWER-SHOVEL OPERATIONS

Investigations of open-cut metal-mining practices have been conducted by engineers of the Bureau of Mines for several years and are continuing. A bulletin is in course of preparation that will discuss the subject in much greater detail than is possible in the present bulletin. The data in table 55 have been compiled for the most part from reports previously published.

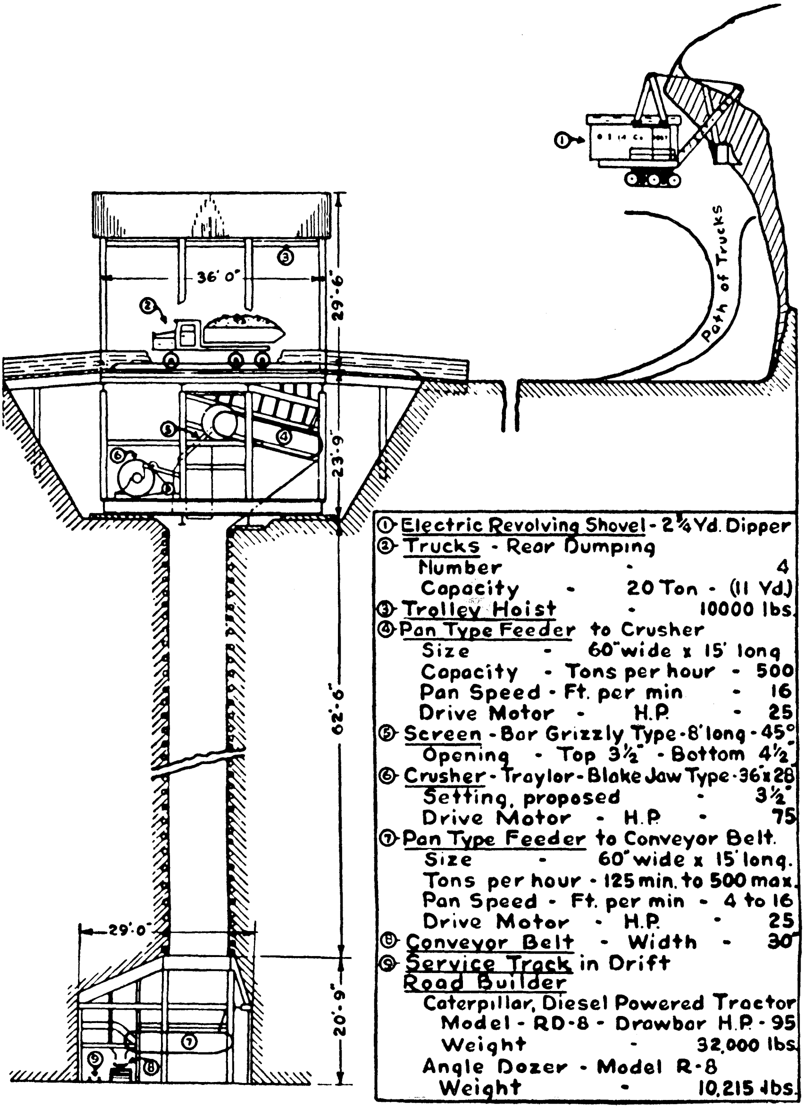

①
Path of Trucks
③
36' 0"
29'-6"
②
⑤
④
⑥
23'-9"
62'-6"
29'-0"
20'-9"
⑦
⑨
⑧
① Electric Revolving Shovel - 2¼ Yd. Dipper
② Trucks - Rear Dumping
Number - 4
Capacity - 20 Ton - (11 Yd.)
③ Trolley Hoist - 10000 lbs.
④ Pan Type Feeder to Crusher
Size - 60" wide x 15' long
Capacity - Tons per hour - 500
Pan Speed - Ft. per min - 16
Drive Motor - H.P. - 25
⑤ Screen - Bar Grizzly Type - 8' long - 45°
Opening - Top 3½" - Bottom 4½"
⑥ Crusher - Traylor - Blake Jaw Type - 36"x 28"
Setting, proposed - 3½"
Drive Motor - H.P. - 75
⑦ Pan Type Feeder to Conveyor Belt.
Size - 60" wide x 15' long.
Tons per hour - 125 min. to 500 max.
Pan Speed - Ft. per min - 4 to 16
Drive Motor - H.P. - 25
⑧ Conveyor Belt - Width - 30"
⑨ Service Track in Drift
Road Builder
Caterpillar, Diesel Powered Tractor
Model - RD-8 - Drawbar H.P. - 95
Weight - 32,000 lbs.
Angle Dozer - Model R-8
Weight - 10,215 lbs.

FIGURE 140.—Spruce truck-dumping station.

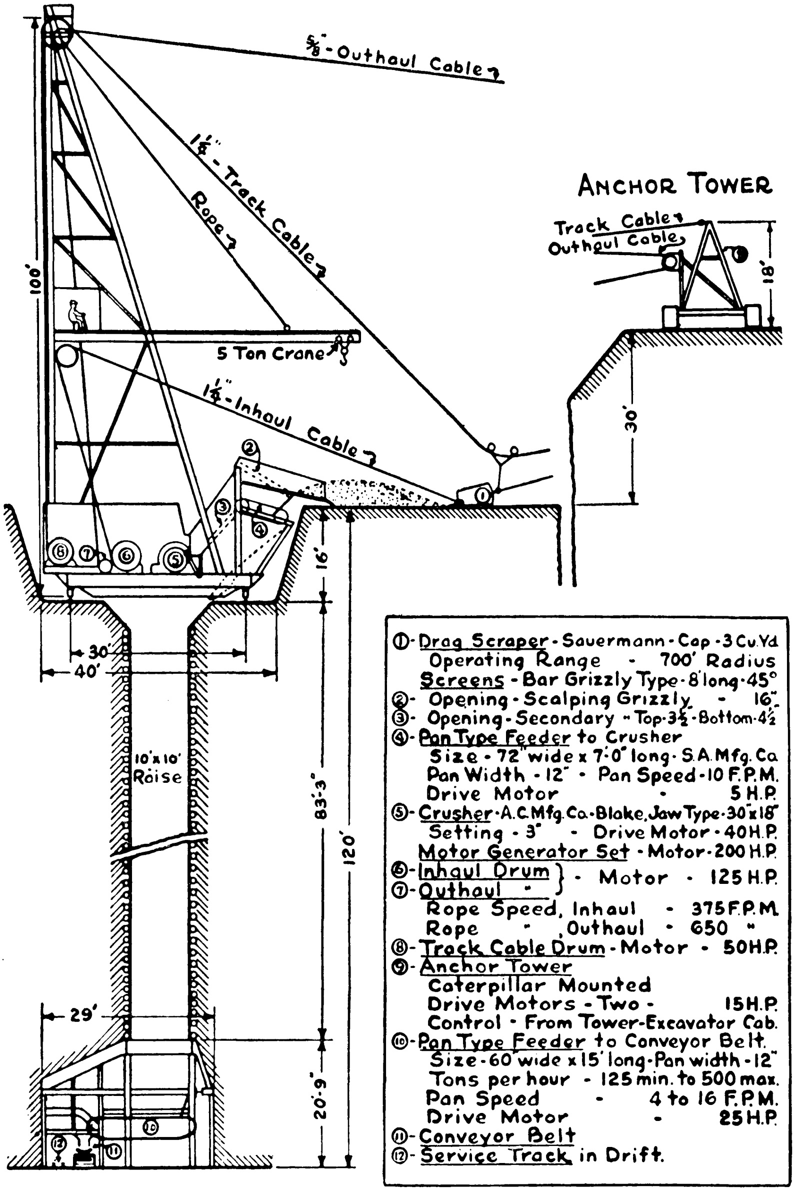

1¼"-Outhaul Cable
1¼"-Track Cable
Rope
5 Ton Crane
1¼"-Inhaul Cable
Anchor Tower
Track Cable
Outhaul Cable
100'
18'
30'
16'
83'-3"
120'
20'-9"
30'
40'
29'
10'x10' Raise
①-Drag Scraper - Sauermann - Cap - 3 Cu. Yd
Operating Range - 700' Radius
Screens - Bar Grizzly Type - 8' long - 45°
②- Opening - Scalping Grizzly - 16"
③- Opening - Secondary - Top - 3½ - Bottom - 4½
④-Pan Type Feeder to Crusher
Size - 72" wide x 7'-0" long - S.A. Mfg. Co.
Pan Width - 12" - Pan Speed - 10 F.P.M.
Drive Motor - 5 H.P.
⑤-Crusher - A.C. Mfg. Co. - Blake, Jaw Type - 30"x18"
Setting - 3" - Drive Motor - 40 H.P.
Motor Generator Set - Motor - 200 H.P.
⑥-Inhaul Drum } - Motor - 125 H.P.
⑦-Outhaul " }
Rope Speed, Inhaul - 375 F.P.M.
Rope " , Outhaul - 650 "
⑧-Track Cable Drum - Motor - 50 H.P.
⑨-Anchor Tower
Caterpillar Mounted
Drive Motors - Two - 15 H.P.
Control - From Tower-Excavator Cab.
⑩-Pan Type Feeder to Conveyor Belt.
Size - 60" wide x 15' long - Pan width - 12"
Tons per hour - 125 min. to 500 max.
Pan Speed - 4 to 16 F.P.M.
Drive Motor - 25 H.P.
⑪-Conveyor Belt
⑫-Service Track in Drift.
①
②
③
④
⑤
⑥
⑦
⑧
⑩
⑪
⑫

FIGURE 141.—Spruce tower-excavator station.

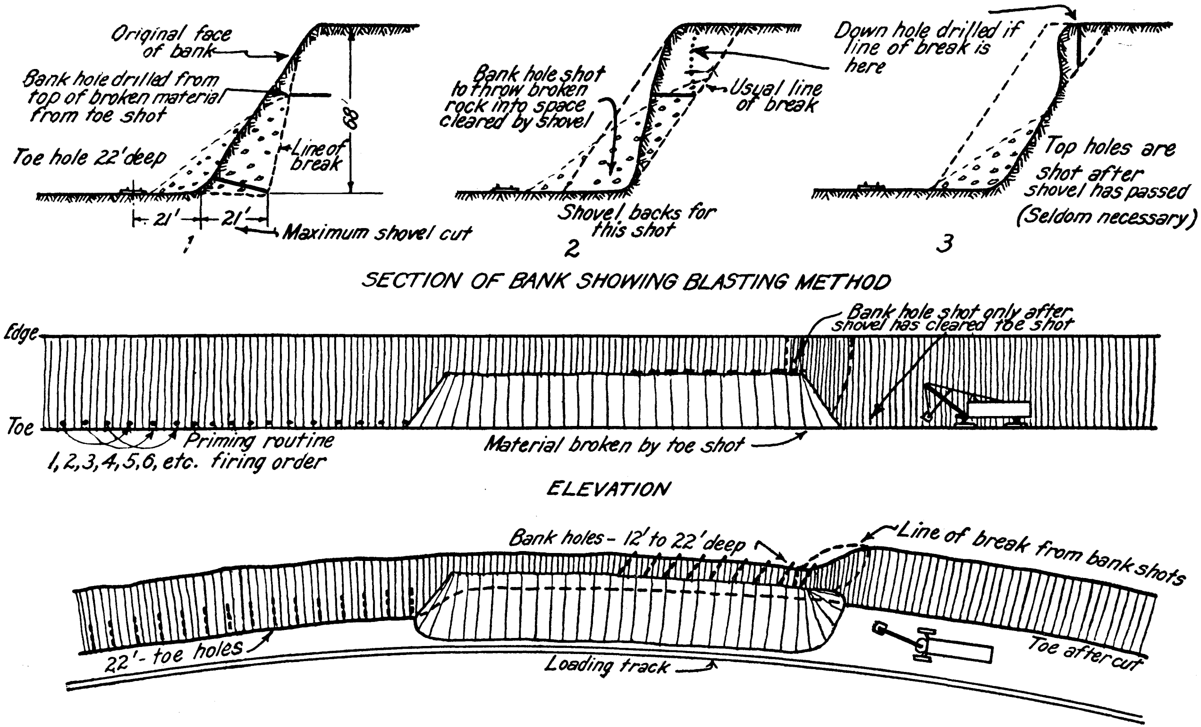

Original face of bank
Bank hole drilled from top of broken material from toe shot
Toe hole 22' deep
Line of break
88
21'
21'
Maximum shovel cut
1
Bank hole shot to throw broken rock into space cleared by shovel
Usual line of break
Shovel backs for this shot
2
Down hole drilled if line of break is here
Top holes are shot after shovel has passed (Seldom necessary)
3
SECTION OF BANK SHOWING BLASTING METHOD
Edge
Toe
Bank hole shot only after shovel has cleared toe shot
Priming routine
1, 2, 3, 4, 5, 6, etc. firing order
Material broken by toe shot
ELEVATION
Bank holes - 12' to 22' deep
Line of break from bank shots
22'- toe holes
Loading track
Toe after cut

FIGURE 142.—Drilling and blasting practice, Utah Copper mine, Utah.

TABLE 55.—*Data on power-shovel operations*

1. GENERAL DATA

Mine and reference	Overburden or capping		Ore body				Ultimate waste-ore ratio	Ultimate over-all pit-wall slope, degrees	Standard benches, feet	
	Kind	Average depth, feet	Kind	Dimensions, feet					Width	Height
				Length	Width	Depth				
Utah Copper, Utah [1]	Leached monzonite porphyry.	115	Easily broken monzonite porphyry.	[2] 6,000	[2] 4,000	[2] 2,000	[3] 1.1-1	26-40	100	40-70
United Verde, Ariz.:[4]										
Upper pit	Diorite greenstone, porphyry, schist.		Oxidized schist	1,200	850	300±	[5] 2.4-1			45-55
Lower pit	do		Schist, massive sulfides	1,000	800	400	[5] 2.4-1	45	Variable	25-35
Chino, N. Mex	Leached granodiorite and quartz diorite.	75±	Hard, tough granodiorite and quartz diorite.	4,000	2,500	425	1-1	45		50
New Cornelia, Ariz	Monzonite		Hard monzonite	3,500	2,000	425	0.8-1	45	100-350	25-35
Sacramento, Ariz.[6]	Porphyry	280	Hard porphyry	1,200 max.	800 max.	227	11 cu. yds. to 8.5 tons ore	45		60, 45, 30
Mesabi iron mine No. 1	Glacial drift		Sandy hematite and limonite	2,600	1,300	245		60		30
Mesabi iron mine No. 2	do	35	Hematite and limonite	2,600	2,000	200				
Mesabi iron mine No. 3	do	70	Sandy hematite	5,200	2,600	230		45		30
Mesabi iron mine No. 4	do	30-60	Hematite and limonite	1,300	700	200				20
Mesabi iron mine No. 5	do		do	(7)						
Mesabi iron mine No. 6	do	110	Sandy hematite and limonite	4,000	4,000					40 max.
Mesabi iron mine No. 7	do	70	do	5,000	1,400	175				40-60
Mesabi iron mine No. 8	do		Hematite and limonite	9,000	1,300	160		60		25
Mesabi iron mine No. 9	do	40	Sandy ore and shipping ore	2,300	1,900	160			60	40 max.
Mesabi iron mine No. 10	do	50	Hematite and limonite	2,600		450			40	30
Weepah, Nev.[8]	Schist, decomposed		Silicified, sheared dike, decomposed. Gold ore.	700	120	70	1.3-1			
Orogrande-Frisco, Idaho [9]		0	Silicified schist and pegmatite; low-grade gold ore.	400	400	250 max.	0			
Oronogo, Mo., 1938 [8]	Limestone and shale		Lead and zinc carbonates and sulfides in dolomite and chert.[10]	750	750	165 max.; 220 ultimate.			22	30 max.
Sulphur Bank, Calif., 1936	Basalt and altered basalt	30				50				
Morenci, Ariz., stripping, 1938.[11]	Porphyry, stripping area 8,000 by 4,500 feet.	216 (500 max.)	Porphyry	6,800	3,800		1.04-1	45	100	50

[1] Soderberg, A., Mining Methods and Costs of the Utah Copper Co., Bingham Canyon, Utah: Bureau of Mines Inf. Circ. 6234, 1930, 23 pp.

[2] Maximum.

[3] Ratio to 1930.

[4] Alenius, E. M. J., Methods and Cost of Stripping and Mining at the United Verde Open-Pit Mine, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6248, 1930, 34 pp.

[5] Lower and upper pits combined.

[6] Ziesmer, H. M., and Mieyr, George, Steam-Shovel Operation at Bisbee, Ariz.: Trans. Am. Inst. Min. and Met. Eng., vol. 68, 1923, pp. 251–269.

[7] Clean-up operation.

[8] Field notes.

[9] Lorain, S. H., Mining and Milling in Idaho County, Idaho: Bureau of Mines Inf. Circ. 7039, 1938, pp. 37–43.

[10] Old underground mine pillars.

[11] Lawson, Walter C., Preliminary Stripping of the Morenci Open Pit, Arizona: Am. Inst. Min. and Met. Eng. Tech. Pub. 980, September 1938.

TABLE 55.—*Data on power-shovel operations*—Continued

2. DRILLING AND BLASTING

Mine and year	Drilling				Blasting			
	Types of drills	Size of holes		Cost per foot of hole	Method	Explosive used		Explosive per ton broken, pounds
		Diameter, inches	Average depth, feet			Springing	Main blasts	
Utah Copper, Utah, 1928	Piston	3¾–2¼	22	$0.025	(See fig. 142) holes sprung.	60 percent ammonia dynamite.	60 percent ammonia dynamite.	0.125
United Verde, Ariz., Upper pit, 1925.	Churn	6	35	4.54 (1926)	Special methods for hot ground.	35 and 60 percent gelatin dynamite.	35 and 60 percent gelatin dynamite.	[13].648
	Leyner	2¾–1 5/16	21–30	[12].42	Vertical holes			
	Hammer	2¼–1 3/16	18–20	[12].42	Toe holes, all holes sprung, some coyote blasting.			
Lower pit: 1928	Churn	6	35		Vertical holes	35 percent gelatin		[13].483
	Leyner on tripods	2¾–1 5/16	20		Toe holes	35 to 50 percent gelatin		
	Hammer	2¼–1 3/16	20		Toe holes, all holes sprung.			
1935	do	2¼–1 3/16	20	.174 per cu. yd.	do	35–60 percent gelatin		.35
Chino, N. Mex., 1929	Churn	4½ and 6	30–55	.62	Vertical holes	40 percent ammonia dynamite.	35 and 40 percent granulated ammonia dynamite.	.167
	Hammer	3¾–1⅜	22		Toe holes, all holes sprung.			
New Cornelia, Ariz., 1930	Hammer suspended from tripods.[14]	2 7/16	28		Vertical holes, sprung	60 percent gelatin dynamite.	Free-running quarry powder.	.17
Sacramento, Ariz., 1921	Churn	6	42	.55	Vertical holes, sprung	40 percent gelatin	Granulated ammonia dynamite.	.40
	Piston	3¼–1½	21	.25	Toe holes, sprung			
Orogrande - Frisco, Idaho, 1937.	Jackhammer		Flat holes		−15° holes sprung	40 percent gelatin dynamite.	Bag powder	.125 (bag)
Oronogo, Mo., 1938	Hammer on tripods.				Toe holes		40 gelatin dynamite	
Sulphur Bank, Calif., 1936	Jackhammers	2⅜–1¾	10		do	40 percent gelatin dynamite.	Hercomite No. 4	.70
Morenci stripping, Ariz. 1938.	do			.147		Deep holes sprung		
	Churn (electrically driven).	9	55±(8 to 10 ft. below grade).	.36 (85 ft. per shift)		Not sprung	Bag powder with gelatin for primer; Cordeau fuse.	

Mesabi iron mine No. 1, 1937.	Churn	6	35	[15] 55	Vertical holes, electric blasting.	None	25 percent gelatin dynamite.	.18
Mesabi iron mine No. 2, 1937.	do	6	26	[15] 80	do	35 percent gelatin dynamite.	Ammonia dynamite	.127
Mesabi iron mine No. 3, 1937.	do	6	30		do		25 percent gelatin dynamite.	.18
Mesabi iron mine No. 4, 1937.	do	6		[15] 20–30	Vertical holes, electric blasting; holes sprung.	35 percent gelatin dynamite.	Ammonia dynamite	.13
Mesabi iron mine No. 5, 1937.	do	6		[15] 150				
Mesabi iron mine No. 6, 1937.	do	6	40 maximum	[15] 110 ore; 25 rock	Vertical holes sprung once in ore.		25 percent ammonia dynamite.	
Mesabi iron mine No. 7, 1937.	do	6		[15] 60	Vertical holes			
Mesabi iron mine No. 8, 1937.	Churn	6		[15] 80	do		25 percent gelatin dynamite in ore.	.09 (ore)
	And air drills			[15] 28			40 to 60 percent in rock	
Mesabi iron mine No. 9, 1937.	Churn	6		[15] 35	do			
Mesabi iron mine No. 10, 1937.	do	6	35	[15] 50	Vertical holes		Gelatin dynamite, 50 to 80 percent in rock 50 percent in ore.	
Weepah, Nev., 1936–37	do				do			

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[12] Labor only.

[13] Per cubic yard.

[14] Later replaced by wagon drills.

[15] Represents drilling speed per 8 hours, in feet.

TABLE 55.—*Data on power-shovel operations*—Continued

3. SHOVEL LOADING

Mine and year	Shovel data					Performance	
	Number	Mounting	Power	Dipper size, cubic yards	Boom length, feet	Material loaded per shovel shift, tons or cubic yards	
						Ore	Waste
Utah Copper, Utah: 1928	8	Caterpillar	A. C. electric	4½	30	3,886 tons	3,692 tons.
1934	15	do	D. C. electric	4½	30	3,989 tons 5,200 tons	3,752 tons.
United Verde, Ariz., Upper pit, 1925	1	Traction wheels, full revolving	Steam	8		1,000 cubic yards	1,000 cubic yards.
	7	Caterpillar [16]	do	4		784 cubic yards	784 cubic yards.
	1	do	do	¾			
Lower pit: 1928	4	Full revolving, caterpillar	Electric	1¾		1,050 tons	1,050 tons.
1935	2	do	do	4			
	1	do	do	3			
	4	do	do	1¾			
Chino, N. Mex., 1929	7	Caterpillar	D. C. electric	4		1,190 cubic yards (1929) 2,810 cubic yards (1931)	1,190 cubic yards (1929). 2,810 cubic yards (1931).
	1	do	do	8			
	3	do	Steam	4			
	1	do	Diesel	1¼			
New Cornelia, Ariz., 1930	10	do	Steam	4	30	1,322 tons	1,322 tons.
	2	do	Electric	4½			
Sacramento, Ariz.	7	Railroad	Steam	3½		1,038 cubic yards	1,038 cubic yards.
Mesabi iron mine No. 1, 1937	1	Caterpillar, full-revolving	Electric	3¼	29	1,000 tons [17]	
	2	Railroad-type	Steam	5	29		
	1	do	do	2½			
Mesabi iron mine No. 2, 1937	1	Caterpillar	Electric	1¾	29.5	2,500 tons [17]	
	3	do	do	3			
Mesabi iron mine No. 3, 1937	2	do	do	4		1,900 [17]	
	1	do	Gasoline	1¼			
Mesabi iron mine No. 4, 1937	2	do	Electric	3	125	1,900 [17]	
Mesabi iron mine No. 5, 1937	2	do	Gasoline	1¼		400 [17]	
Mesabi iron mine No. 6, 1937	1	do	Electric	8	90		
	3	do	do	5	28		
	2	do	do	4			
	1	do	do	2			
	1	do	Diesel-electric	1⅞			
Mesabi iron mine No. 7, 1937	1	do	Electric	4			
	1	do	do	2¼			
	1	do	Steam	4			
	1	do	do	6			
Mesabi iron mine No. 8, 1937	1	do	Electric	4			
	2	do	do	5			
	3	do	Steam	5			
	3	1 caterpillar, 2 railroad	do	4			

Mesabi iron mine No. 9, 1937	3	do	Electric	4		4,000[17]	
	1	do	do	8			
Mesabi iron mine No. 10, 1937	1	do	do	4		2,500[17]	
	1	do	do	8			
Weepah, Nev., 1936–37	1	do	Gasoline	1¼		300	
Orogrande-Frisco, Idaho, 1937	1	do	Diesel	1		500–600 tons, work only 1 shift	
Oronogo, Mo., 1938	1	do	Gasoline, 90 h. p.	¾		1,800 maximum (9 hours)	
Sulphur Bank, Calif., 1936	2	do	Diesel, 80 h. p.	1¼	28	35 tons	187 tons.
	1	Bulldozer	Diesel				
Morenci stripping, Ariz., 1938	4	Full revolving caterpillar	Electric Ward Leonard control	3–4½			5,000 tons.
	1	Bulldozer to each shovel		1–5½			

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[16] Railroad type converted to caterpillar mounting.

[17] Long-ton capacity in 8 hours.

TABLE 55.—*Data on power-shovel operations*—Continued

4. HAULAGE

Mine and year	Average 1-way haul, miles	Maximum grade, percent	Maximum curvature, degrees	Locomotives and trucks			Cars			
				Number	Weight, tons	Power	Number	Capacity, tons or cubic yards	Type	Number in train
Utah Copper, Utah 1928	3.0	[18] 4		41 locomotives.	75	Electric		Ore, 80–90 tons Waste, 30 cubic yards	Railroad, ore Air-dump, waste	Ore, 11 Waste, 5
United Verde, Ariz.: Upper pit, 1925.	0.8	[18] 3	24	5 locomotives. 2 locomotives.	82. 5 53. 5	Steam do	30	25 cubic yards	Air-dump	6 loaded 4 loaded
Lower pit, 1928		12		17 trucks	3-ton body.	Internal combustion.	17 trucks	10 tons (trucks)	Side-dump trucks	
Chino, N. Mex., 1929	2.7	[18] 2. 5	[19] 20	23	90	Steam	110+	20 and 30 cubic yards	Automatic air-dump.	(20)
New Cornelia, Ariz., 1930	1.5 ore; 1.8 waste	[18] 2	[19] 20	15	67	do	82	20 cubic yards	do	5 or 6
Sacramento, Ariz., 1921		2½	[21] 40	15	53–54	do	5 45	do 25 cubic yards	do do	
Mesabi iron mine No. 1, 1937	1, stripping 3¼ ore.	2	30	6	65	do	42	20 cubic yards	do	6
Mesabi iron mine No. 2, 1937	1.5 ore; 0.75 waste.	[18] 2½	28	5 1	80 90	do do		50 and 75 tons ore 20 cubic yards waste	Railroad Air-dump	5–4
Mesabi iron mine No. 3, 1937	6	[18] 2½	[19] 12	5	65	do		30 cubic yards	do	4
Mesabi iron mine No. 4, 1937		[18] 3	26	3		do		50 tons 20 cubic yards	Railroad Air-dump	4 3
Mesabi iron mine No. 6, 1937	4	[18] 2	24	8	90	Electric		75 tons 20 and 30 cubic yards	Railroad Air-dump	5 6
Mesabi iron mine No. 7, 1937	2.5	[18] 2½	[19] 16	4 2	65 45	Steam do		50–75 tons 30 cubic yards	Railroad Air-dump	4 4
Mesabi iron mine No. 8, 1937	2.75	[18] 1. 86	[19] 15	2 6 4	65 85 125	do do do		75 tons 25 cubic yards	Railroad Air-dump	12
Mesabi iron mine No. 9, 1937	1.75	[18] 3	[19] 16	7	60	Electric		50 tons 30 cubic yards	Railroad Air-dump	4
Mesabi iron mine No. 10, 1937	5 ore; 7 waste	[18] 2¼	[19] 18	3	109	Steam		50 tons 30 cubic yards	Railroad Air-dump	4 5
Weepah, Nev., 1936–37	0.25			2 trucks		Gasoline		Trucks		
Orogrande-Frisco, Idaho, 1937	Short			2 trucks			2	5-ton trucks		
Oronogo, Mo., 1938	⅛	15; average 12½		Trucks		Gasoline	3 plus, 1 spare.	1½-ton trucks		
Sulphur Bank, Calif., 1936	⅛ waste; ½ ore	17		4 1		do Diesel		6 tons 10 tons		
Morenci stripping, 1938	0.6	Down grade to date.		18 trucks		Gasoline, 130 and 150 h. p.		Trucks, 22½ cubic yards 33 tons pay load.		

5. OPERATING COSTS. A

Mine and year	Stripping and waste inclusions						Ore mining					
	Cubic yards	Costs per cubic yard					Tons	Cost of mining per ton				
		Drilling and blasting	Loading	Haulage	Other	Total		Drilling and blasting	Loading	Haulage	Other	Total
Utah Copper, Utah:												
1928	7, 220, 034	$0. 060	$0. 042	$0. 145	$0. 026	$0. 273	16, 558, 500	$0. 027	$0. 019	$0. 053	$0. 018	$0. 117
1934	-------	-------	-------	-------	-------	-------	4, 086, 800	. 026	. 020	. 040	. 022	. 108
United Verde, Ariz.:												
Upper pit, 1925	1, 647, 469	. 241	. 214	. 211	. 059	. 725	272, 005	. 149	. 117	. 104	-------	. 370
Lower pit, 1928	322, 538	. 293	. 161	. 198	. 018	. 670	731, 488	. 139	. 077	. 086	-------	. 302
Chino, N. Mex., 1929:												
Steam shovels	1, 577, 373	. 107	. 192	. 284	. 049	. 632	1, 335, 476	. 051	. 079	. 131	. 017	. 278
Electric shovels	2, 976, 357	. 097	. 078	. 219	. 030	. 424	2, 621, 340	. 049	. 036	. 108	. 014	. 207
New Cornelia, Ariz., 1930	[22] 1, 040, 610	-------	-------	-------	-------	-------	2, 376, 764	[23]. 070	[23]. 049	[23]. 066	[23]. 025	[23]. 210
Sacramento, Ariz., 1921	-------	[24]. 202	[24]. 130	[24]. 200	[24]. 016	[24]. 548	-------	-------	-------	-------	-------	-------
Mesabi iron mine No. 1	-------	-------	-------	-------	-------	-------	[25] 679, 590	. 035	. 048	[26]. 110	. 139	. 332
Weepah, Nev., 1936–37, 6 months	63, 348 (ore and waste)	. 122	. 106	. 092	[27]. 106	. 343	-------	-------	-------	-------	-------	-------
Orogrande-Frisco, Idaho, 1937	-------	-------	-------	-------	-------	-------	128, 671	. 068	. 062	-------	. 066	[28]. 196
Oronogo, Mo., 1938	150, 000	-------	-------	-------	-------	. 160	50, 000	-------	-------	-------	-------	. 160
Sulphur Bank, Calif., 1936	103, 843 (approx.)	-------	-------	-------	-------	-------	19, 436	-------	-------	-------	-------	-------
Morenci stripping, 1938	-------	[29]. 0106	-------	[30]. 038	-------	-------	-------	-------	-------	-------	-------	-------

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[18] Compensated for curvature.

[19] On main lines.

[20] 200 cubic yards on favorable grades, 120 to 140 cubic yards on adverse grades.

[21] Some 65° in early operations.

[22] Tons.

[23] Per ton ore and waste.

[24] Per cubic yard ore and waste.

[25] Long tons of 2,240 pounds.

[26] Includes track expense, $0.021.

[27] Operating bulldozer.

[28] Includes taxes, depreciation, and diamond drilling.

[29] Per ton, based on shovel operation and maintenance cost of $52.81 per 8-hour shift and 5,000 tons per shift.

[30] Per ton.

TABLE 55.—*Data on power-shovel operations*—Continued

5. OPERATING COSTS. B

Mine and year	Stripping, per cubic yard						Ore mining, per ton					
	Labor and super-vision	Com-pressed air, drills, and steel	Power and fuel	Explo-sives	Other supplies and general	Total	Labor and super-vision	Com-pressed air, drills, and steel	Power and fuel	Explo-sives	Other supplies and general	Total
Utah Copper, Utah:												
1928	$0.127	$0.006	$0.026	$0.034	$0.081	$0.274	$0.044	$0.008	$0.012	$0.015	$0.038	$0.117
1934	-----	-----	-----	-----	-----	-----	.035	-----	.005	.013	-----	-----
United Verde, Ariz.:												
Upper pit, 1925	.248	.036	.092	.087	.262	.725	.145	.023	.038	.046	.118	.370
Lower pit, 1928	.222	.081	.014	.079	.274	.670	.100	.037	.006	.035	.124	.302
Chino, N. Mex., 1929:												
Steam shovels	.324	.012	.050	.048	.198	.632	.148	.006	.021	.023	.080	.278
Electric shovels	.227	.012	.012	.044	.129	.424	.114	.006	.005	.021	.060	.207
New Cornelia, Ariz.: 1930	-----	-----	-----	-----	-----	-----	[23].125	[23].001	[23].034	[23].025	[23].025	[23].210
Mesabi iron mine No. 1	-----	-----	-----	-----	-----	-----	[25].212	(25)	[25].068	[25].019	[25].033	[25].332
Mesabi range; all open-pit mines, 1926 to 1935, inclusive [31]	-----	-----	-----	-----	-----	[30].274	.124	.287				[32].411
Orogrande-Frisco, Idaho, 1938	-----	-----	-----	-----	-----	-----	.090	.106				.196
Sulphur Bank, Calif., 1936	[23].169	-----	[23].028	[23].014	[23].142	[23].353	-----	-----	-----	-----	-----	[33]2.24

6. COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Mine and year	Man-hours per ton or per cu. yd.							Explo-sives, pounds per ton or cu. yd.	Power Kw-hr., per ton or cu. yd.	Fuel units per ton or cu. yd.
	Drilling and blasting	Loading	Haulage	Track main-tenance	Repairs	Miscel-laneous	Total			
Utah Copper, Utah, 1929:										
Stripping, per cubic yard	0.032	0.028	0.057	0.122	0.084	0.028	0.351	0.241	1.45	0.003 ton coal.
Mining, per ton	.014	.011	.024	.031	.025	.013	.118	.105	.79	
United Verde, Ariz.:										
Upper pit, 1925, ore and stripping per cu. yd	-----	-----	-----	-----	-----	-----	.678	.648	-----	
Lower pit, 1928, ore and stripping per cu. yd	-----	-----	-----	-----	-----	-----	.606	.483	-----	
Chino, N. Mex., per ton	-----	-----	-----	-----	-----	-----	-----	.165	[34].04	
New Cornelia, Ariz., 1930, ore and stripping, per ton	-----	-----	-----	-----	-----	-----	.190	.168	1.79	0.016 barrel fuel oil.
Mesabi iron mine No. 1	.014	.032	.083		.102		.231	.180	1.78	0.0053 ton coal.
Sulphur Bank, Calif., 1936	-----	-----	-----	-----	-----	-----	[35]1.90 [36].300	.700	-----	

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[23] Per ton ore and waste.

[25] Long tons 2,240 pounds.

[30] Cost of stripping and mining per ton of ore mined.

[31] Craig, John J., Mining Directory of Minnesota, 1937: Bull. University of Minnesota, vol. 40, No. 39, May 1, 1937, p. 226.

[32] Stripping, $0.274, and royalties, $0.449, gives total production cost of $1.134.

[33] Cost of stripping and mining per ton of ore.

[34] Power for shovels only.

[35] Per ton of ore for stripping and ore mining.

[36] Per ton of ore and stripping.

ORE DRESSING

INTRODUCTION

In this bulletin the subject of ore dressing is approached from the economic rather than from the technologic standpoint, and the supporting data have been compiled with a view to their bearing upon the economic exploitation of ore deposits. Results obtained are stressed rather than the details of processes and the physics and chemistry involved. For discussions of the latter, the reader is referred to numerous publications prepared by the Metallurgical Division of the Bureau of Mines, textbooks on ore dressing, the technical journals, and publications of the technical societies.

The object of ore dressing is, broadly, to prepare the crude ore for more economical transportation and for further processing to recover the metallic contents. Although perhaps not ore dressing in the strict sense of the term, the direct extraction of the metals (especially the precious metals) from the ores also is considered here as an ore-dressing or a "milling" process. Preparation by ore dressing includes the concentration of the valuable minerals within a smaller bulk than that of the original ore by removal of worthless constituents, the separation from each other of the different valuable minerals to give separate products in each of which the minerals of one metal predominate, or the removal of constituents that have a deleterious effect in further processing and thus result in a penalty charge by the smelter.

By concentrating the metal contained in the crude ore through removal of valueless constituents, the gross weight is reduced to a fraction of that of the original ore, thus lowering the transportation cost per unit of contained metal. Many ores that could not possibly be worked if shipped in the crude state can be worked profitably if concentrated before shipment.

In commercial concentration processes some ore mineral is necessarily lost in the tailing or reject, the percentage so lost ranging between wide limits, depending on the character of the ore and the refinements of the ore-dressing methods employed. The question of how far to go with ore-dressing refinements to effect a high percentage of recovery of the valuable constituents of the ore is an economic one. Each step in the concentration of an ore adds to the cost of the operation, and unless the value of the additional mineral recovered exceeds the extra cost of recovering it, the refinement is obviously uneconomical.

Complex ores containing minerals of two or more metals frequently can be exploited economically only if the different metals are separated before smelting. For example, lead ores or concentrates are charged a penalty by the smelter if they contain more than a certain percentage of zinc [84]; and if the zinc content cannot be reduced below a certain limit, the penalty, which increases with increase in zinc content, may be so great that it would be unprofitable to operate. In this connection it is noted that many complex western ores could not be worked profitably and were left unmined for years even though high in lead, zinc, and silver, before concentration by differential flotation was developed. By means of differential flotation it became possible to make a high-grade lead concentrate containing very little zinc and a high-grade zinc concentrate containing little lead, both of which could be shipped profitably.

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[84] Gardner, E. D., and Allsman, Paul T., Open Schedules for Gold and Silver Ores and Concentrates at Western Custom Smelters: Bureau of Mines Inf. Circ. 6926, 1936, 25 pp.

Table 56 is reproduced from Information Circular 6926 to illustrate the charges made by smelters for excess deleterious elements.

TABLE 56.—*Schedules for gold and silver ores at lead smelters (July 1936)*

Plant	Payments												
	Gold [1]			Silver [2]			Lead [3]			Copper [4]			Iron and lime
	Minimum paid for, ounce per ton	Payments		Minimum paid for, ounce per ton	Minimum deductions, ounce per ton	Percent paid for	Deduction		Percent of quotation after deduction	Deductions		Percent of quotation after deduction	
		Classes of ore, ounces per ton	Rate per ounce				Units	Cents per pound		Pounds per ton	Cents per pound		
El Paso, Tex	0.03	All	$32.81825		0.5	[5] 95	1.5	1.425	[6] 90	8	5.025	[7] 95	
Murray, Utah	.02	All	31.81825	1.0	.5	95	1.5	1.5	[6] 90	15	5.5	100	Iron.[10] Lime.[11]
East Helena, Mont.[13]	.03	0.03 to 3 3–5 5–10 Over 10	31.81825 32.31825 32.6743 33.0304		1.0	95	1.5	1.5	[6] 90	20	6.0	[14] 100	Iron.[15]
Selby, Calif.[16]	.03	Under 5 5–15 Over 15	31.81663 32.31663 32.81663	1.0	1.0	95			0			0	
Do.[18]	.03	Under 5 5–15 Over 15	31.81663 32.31663 32.81663	1.0	1.0	95			0			0	
Leadville, Colo.[20]	.03	0–1 Over 1	31.81825 32.31825		1.0	95	1.5	0	90	20	6.5	100	Iron.[21]
Do.[22]	.03	0–1 Over 1	31.81825 32.31825		1.0	95	1.5	1.5	[23] 100	20	6.5	[24] 100	Iron.[25] Lime.[26]
Do.[29]	.03	0–1 Over 1	31.81825 32.31825		1.0	95	1.5	1.5	90	20	6.5	100	
Midvale, Utah	.02	0–5 Over 5	31.81825 (30)	1.0	.5	95	1.5	1.5	[31] 90	[32] 15	5.5	90	Iron.[33]
Kellogg, Idaho [36]	.05	Under 5 5–10 Over 10	31.81825 32.17431 32.53037	1.0		[37] 95	1.25	0	[6] 90	20	8.0	[38] 100	
Do.[41]	.05	Under 5 5–10 Over 10	31.81825 32.17431 32.53037	1.0		[37] 95			0	20	8.0	[38] 100	

See footnotes at end of table.

Table 56.—*Schedules for gold and silver ores at lead smelters (July 1936)*—Continued

Plant	Deductions																	
	Treatment charge			Penalties														
				Insoluble		Zinc		Arsenic (As), antimony (Sb), tin (Sn)			Bismuth		Sulfur			Moisture		
	Gross value, dollars	Base per ton	Maximum	Units free	Charge per unit	Units free	Penalty per unit for excess	Element or combination	Units free	Penalty per unit for excess	Units free	Penalty per unit for excess	Units free	Penalty per unit for excess	Maximum penalty	Units free	Penalty per unit for excess	Maximum penalty
El Paso, Tex	0-25 Over 25	$3.70 (8)	----- $6.70	0	$0.05	5	$0.30	As Sb+Sn	2 1	$0.50 1.50	(9)	$0.50	2	$0.20	$2.00	-----	0	-----
Murray, Utah	(12)	2.50	-----	0	.10	6	.30	As+Sb	2	.50	-----	0	2	.25	2.50	-----	0	-----
East Helena, Mont.[13]	0-30 30-40 Over 40	6.00 5.50 5.00	----- ----- 6.00	-----	0	5	.30	As Sb	2 1	.50 2.00	(9)	.50	-----	0	-----	-----	0	-----
Selby, Calif.[16]	All	6.00	6.00	(17)	0	-----	0	As Sb Sn	1	.50	0	.50	-----	0	-----	-----	0	-----
Do.[18]	0-35 Over 35	6.50 (19)	----- 10.00	-----	0	-----	0	As Sb Sn	1	.50	0	.50	-----	0	-----	-----	0	-----
Leadville, Colo.[20]	All	6.00	6.00	-----	0	5	.50	As Sb Sn	.5	1.00	.05	.50	-----	0	-----	10	$0.05	0
Do.[22]	0-8 8-10 10-50	4.00 (27) (28)	----- ----- -----	0	.5	8	.30	As Sb Sn	.5	1.00	.05	.50	1	.25	2.50	10	.05	0
Do.[29]	All	8.50	8.50	-----	0	8	.30	As Sb Sn	.5	1.00	.05	.50	-----	0	-----	10	.05	0
Midvale, Utah	(34)	2.50	-----	0	.10	6	.30	-----	-----	0	(35)	-----	2	.25	2.50	-----	0	-----
Kellog, Idaho[36]	(39)	12.00	-----	-----	0	5	.30	As Sb	0	1.00	0	2.50	4	[40].25	2.00	6	.20	$2.00
Do.[41]	0-20 (42)	6.50 -----	----- 9.00	-----	0	5	.30	As Sb	0	1.00	0	2.50	4	.25	2.00	6	.20	2.00

[1] Payments on gold are based on the new mint price of $35 per ounce (net $34.9125).

[2] Payments on silver are based on the new mint price for new American mined ore at 77 cents per ounce.

[3] Payments on lead are based on the New York quotation for common desilverized lead.

[4] Payments on copper are based on the Engineering and Mining Journal quotations.

[5] Less a deduction of 1½ cents per ounce.

[6] Nothing paid for lead less than 5 percent wet assay.

[7] Nothing paid for copper less than ½ percent.

[8] Add 10 percent to the base charge for the excess value over $25 per ton.

[9] 0.1 percent of the wet-lead assay free.

[10] Pay for all at 6 cents per unit.

[11] Pay for all at 5 cents per unit if 5 percent or over.

[12] Add 10 cents to base charge for each unit of lead under 30 percent and deduct 10 cents for each unit of lead over 30 percent.

[13] Siliceous-ore schedule; schedule for ores and concentrates with an excess of iron are identical, except that treatment charge is a flat $5 per ton.

[14] Nothing paid for copper less than 1 percent.

[15] No credit.

[16] Schedule for gold concentrates.

[17] Add 10 cents per ton per unit of iron short of 25 units excess over insoluble.

[18] Schedule for crude siliceous gold ore.

[19] Add 10 percent to the base charge for the excess value over $35 per ton.

[20] 1934 schedule for iron ores and concentrates; applies only on ores and concentrates containing 20 percent or more of iron excess over insoluble.

[21] Excess over insoluble all at 10 cents per unit not to exceed $3 per ton.

[22] Crude-ore open schedule.

[23] When lead is over 9 cents per pound deduct 25 percent of the excess.

[24] When copper is over 15 cents per pound deduct 25 percent of the excess.

[25] Pay for iron plus manganese at 5 cents per unit but credit is not to exceed charge for insoluble.

[26] All at 8 cents per unit if 10 percent or over.

[27] Add 25 percent of gross value to the base treatment charge when it is between $8 and $10.

[28] Add 10 percent of the gross value to the base charge when it is between $10 and $50.

[29] 1934 siliceous ore special for ores with 50 percent excess insoluble.

[30] On direct smelting ores containing over 5 ounces gold per ton the price per ounce for the excess will be to the mutual agreement between the buyer and shipper.

[31] No payment for lead under 3 percent dry assay.

[32] Minimum deduction.

[33] All at 6 cents per unit.

[34] Based on 30 percent dry lead assay. Debit 10 cents for each unit of lead under 30 percent and credit 10 cents for each unit of lead above 30 percent.

[35] The Midvale smelter reserves the right to reject any shipment containing more than 0.1 percent bismuth.

[36] Lead ore open schedule.

[37] Ore over 35 ounces per ton deduct 2 cents per ounce.

[38] No payment for copper under 1 percent or when quotation is 8 cents per pound or less.

[39] Based on 50 percent lead. Add 10 cents per unit when over 50 percent and deduct 10 cents per unit when under 50 percent.

[40] Penalty only applies to ore under 20 percent lead; no penalty for ore 20 percent lead or over.

[41] Siliceous ore open schedule; ores containing no lead or under 5 percent for which no payment is made (lead determined by wet method less a deduction of 1¼ units).

[42] Between $20 and $35, $7.00 per ton; between $35 and $50, $7.50 per ton; between $50 and $75, $8.00 per ton; between $75 and $100, $8.50 per ton; over $100, $9.00 per ton.

Gold and silver associated with base-metal ores usually are recovered in the smelter, where the base metal acts as a collector; later they are separated from the base metals in the refining process. Straight gold and silver ores or those containing only very small percentages of base metals (commonly known as dry siliceous ores) usually can be treated most economically at the mine by processes that result in the direct production of bullion. This avoids the cost of shipping ore and concentrates and payment of smelter charges and deductions, so that the net return to the miner on the gold and silver may range from a few to 15 or more percent higher than the amount that would be received were these metals recovered at the smelter. Sometimes it is profitable to recover part of the gold and silver by amalgamation or cyanidation and also produce a high-grade concentrate containing most of the associated sulfide minerals and the balance of the economically recoverable precious metal. The most profitable procedure will be determined in a given instance by local conditions, among the determining factors being the concentration ratio, the cost of milling and the percentage of recovery that can be made at the mine, the freighting cost, and the smelting charges, including deductions on ores and concentrates. These and other factors will be discussed briefly but in greater detail under Gold Ores.

The minimum grade of ore that can be smelted directly without previous ore-dressing treatment varies, depending largely on the natural fluxing characteristics of the ore, total tonnage of ore, distance to the smelter, haulage and freighting costs, and variations in treatment charges as between different ores and concentrates. An ore that is self-fluxing may be of lower grade for direct smelting than one that requires the addition of a large amount of barren flux. On the other hand, a low-grade ore sometimes may be profitably mined and shipped direct because it is high in silica, iron, or lime, or some other fluxing material required by the smelter. The total tonnage of ore available may be a governing factor, since if it is large enough and fuel and flux likewise are available within a reasonable distance, it may pay to erect a smelter near the mines, whereas if the tonnage is small the returns would not be large enough to amortize the capital outlay for a smelter and pay a profit on the venture. Isolation of the property and consequent high costs for freight obviously raise the cut-off grade between direct smelting and milling ores. Another factor is the concentration-ratio (tons of crude ore per ton of concentrate produced therefrom) and the percentage of recovery by concentration. The higher the concentration ratio without appreciably reducing the recovery, the greater is the advantage in concentration before shipment.

Selection of the best method to employ in a given instance requires not only highly specialized technical knowledge but also a knowledge of relative costs and ability to appraise correctly the various economic factors involved. The economic factors include average price of metals to be recovered, total tonnage to be treated, capital available, water supply, commercial affiliations of the company, royalties on processes and equipment, situation with respect to market and supply bases, transportation facilities, availability of power and fuel, facilities for disposal of tailing, character of labor supply, construction costs, and government regulations.

Ore-dressing methods fall naturally into two general subdivisions—(1) concentration methods and (2) direct-recovery methods (amalgamation and leaching or wet methods).

Concentration methods are employed to remove worthless gangue material and concentrate the valuable minerals of the crude ore in a smaller bulk. Thus, cobbing and hand sorting are crude yet in some instances very efficient concentrating processes. Drying to remove excess moisture in the ore is another simple form of concentration that reduces the relative weight of a worthless constituent of the ore (water) to that of the ore minerals.

Where all or most of the metal is present in the fines resulting from crushing, concentration may be effected by simple coarse crushing and screening, in which case the screen oversize is discarded.

However, the principal concentration methods and those having the widest application to all kinds of ores are gravity and flotation methods. Gravity concentration usually is effected in water or, less often, in a heavy-liquid medium and is based on differences in specific gravity of the ore and the gangue minerals. The formation of placer deposits is an example of natural gravity concentration, and the recovery of placer gold by panning, rocking, and sluicing exemplifies simple forms of artificial gravity concentration. Dry gravity concentration is exemplified by the separation of placer gold and associated heavy minerals from the lighter sands by agitation of the material by means of air pulsations through a riffled, porous deck.

Flotation concentration is effected by causing the selective attachment of a froth to certain minerals in a finely ground ore and agitated with water and frothing reagents. The froth, by its buoyant effect, causes the attached mineral particles to float, whereas the associated gangue minerals comprising the ore will sink. Another important method is magnetic concentration whereby particles of magnetic minerals are removed from a moving ore stream by electromagnets. Electrostatic separation also is employed sometimes, principally in dressing nonmetallic minerals.

Although the best general method of treatment for an ore may be apparent at the outset, it often is not and must be determined and details worked out by preliminary testing in the laboratory, in a pilot plant, or both, if a high extraction and low cost are to be obtained. One can seldom, if ever, visit a commercial ore-dressing plant, no matter how long it has been in operation, in which some changes are not being made or contemplated, unless the source of ore supply is nearly exhausted, which emphasizes the point that careful ore testing should always precede the design of a new plant. Seldom are there two districts in which the ores respond equally well to exactly the same details of treatment, and often the ores from mines in the same district or even those from different sections of the same mine require different treatment to obtain optimum results.

GOLD ORES

The fact that a product of high value and small bulk can be produced at a gold mine, often by relatively simple milling processes, has made it possible to operate gold mines successfully in remote and inhospitable regions where high costs of transportation, fuel, and power would make it impossible to exploit profitably base-metal ores of equal dollar value. As a result, in many instances gold-mine development has been the advance guard of civilization in regions where other mineral resources were developed later and agriculture and industry followed.

TYPES OF GOLD ORES

From the milling viewpoint, gold ores may be grouped as follows:

1. High-grade ores suitable for shipment to smelters or for treatment by direct refinery methods without intermediate concentration.
2. Free-milling ores in which most of the gold may be amalgamated with mercury after suitable grinding, but without roasting, leaching, or other preliminary treatment.
3. Nonamalgamating ores, which, because of intimate association of the gold with pyrite, antimony, or arsenic, combination with tellurium, rusty oxide coatings, or the presence of graphite or greasy substances, must be subjected to treatment by other methods than amalgamation.
4. Ores that will yield part of their gold content by amalgamation but have to undergo additional treatment by other methods to give a satisfactory over-all recovery. Most ores belong to this class, although the distinction between such ores and those of the preceding and following groups is based more often on economic rather than physical considerations.
5. Ores that contain small amounts of amalgamable gold, with most of the gold in other forms, carrying, in addition, secondary amounts of base metals that are worth the expense of recovery.
6. Highly siliceous ores, often of low grade, which are salable under certain conditions to lead or copper smelters for use as flux. They may command a premium for their fluxing value over and above the value of their contained gold.
7. Base-metal ores in which the gold has minor or secondary value. These contribute heavily to total production, but since they are milled in plants designed primarily to recover other metals they are not considered further here.
8. Placer gold. Recovery of metal from this type of material has been touched upon in an earlier section. A combination of gravity concentration and amalgamation generally is resorted to for treating placer material, the gold having been partially concentrated by natural agencies.

METHODS OF TREATMENT

In the milling of gold ores, as with other kinds of ores, the treatment that will give the best metallurgical and economic results in any particular instance is determined primarily by the character of the ore, although other factors previously mentioned must be considered also. The principal ore characteristics that govern the selection of the milling method are grade of ore and uniformity of tenor, the size of the gold particles, the nature of the associated minerals, and the degree to which the gold particles are locked within them. The details of the process also will be determined by these characteristics and to a considerable extent by the crushing and grinding qualities of the ore.

Ores of very low grade usually can be exploited successfully only when treated on a large scale in plants of high capacity, which involve heavy initial capital outlay. Higher-grade ores may be treated profitably in smaller plants with lower initial outlay per dollar of output, unless the ore is refractory and a complex, expensive treatment is unavoidable. Extremely rich ore mined from "pockets" usually is hand-sorted and treated in the mill refinery or is shipped direct to a smelter.

In mills treating rich gold ores it is usually necessary to employ more than one process to recover a high percentage of the gold. The tailing from the first or primary process employed to treat such ores usually carries enough gold to warrant secondary treatment by another process with or without regrinding or by the same process after regrinding. With low-grade ores it may be necessary to employ two or more processes also, the first to produce a product from which a large part of the gangue has been removed and the following ones for extraction of the gold or production of a high-grade concentrate.

Plants for treating ore of uniform grade are the simplest to design and operate, other things being equal. Sudden additions of considerable high-grade ore to an average medium or low-grade mill feed tend to throw the process out of balance and usually will cause increased tailing loss. However, if the high-grade ore is free milling, the feed to the next process may be kept more uniform by first removing much of the gold by amalgamation; or, if the high grade contains coarse gold, by catching it in traps before the ore passes to the main process.

The size of the gold particles in the ore often has considerable influence on the method of treatment to be employed. Because gold is malleable, ordinary grinding will not reduce coarse gold particles to a size that can be dissolved completely by cyanide in the time usually given for solution of fine gold in cyaniding practice, and it is, therefore, desirable to remove coarse gold first by means of traps, blanket tables, jigs, or other devices.

If the gold occurs in extremely small particles, the ore must be ground fine to liberate them from the gangue and expose their surfaces to contact with cyanide solution, or with mercury, as the case may be, or to permit making a high-grade concentrate without carrying into the tailing prohibitive amounts of gold attached to the gangue particles. The fineness of grinding may be determined to some extent by the milling method, which in turn is influenced by the associated minerals, their grindability, and their effect on the applicability of the cyanide, amalgamation, or concentration processes. If the gold is closely associated with sulfides of the base metals, relatively coarse grinding may suffice for gravity or flotation concentration; whereas, if associated with quartz or silicate minerals, much finer grinding may be required to liberate the gold for amalgamation or cyanidation. If the gold is in such fine state and so intimately associated with the gangue minerals, whether sulfides or other minerals, that sliming is necessary to liberate it, cyanidation usually is the best method of recovery to employ unless the ore contains deleterious constituents in quantities sufficient to inhibit solution of the gold, foul the solutions, or react with the cyanide. In some ores of this kind, however, the finely divided gold is contained principally in sulfides that can be concentrated at relatively coarse sizes by flotation, permitting the discard of a large part of the gangue before finer final grinding and treatment of the concentrate by cyanidation.

The nature of the associated ore minerals is one of the most important factors in the selection of treatment methods. Certain constituents of ores may, by causing poor extraction or high operating costs, rule out of consideration a treatment that otherwise would be suitable, or may compel the use of expensive steps that otherwise would not be required.

Pyrite is present in most unoxidized gold ores, where it nearly always contains some gold. Complete liberation of this gold from the pyrite usually entails fine grinding, the cost of which may be prohibitive; and, even if freed, the gold particles often retain a surface film of sulfide capable of preventing amalgamation. Generally it has been found that ores containing much vein pyrite will not yield a satisfactory gold recovery by amalgamation alone but must be treated further by concentration or cyanidation. Where flotation is employed, it may be followed by cyanidation of the concentrates, of the tailing, or both, or by smelting of the concentrates.

Other base-metal sulfides have effects similar to those of pyrite and, in addition, may have a slight tendency to foul mercury, but usually only the sulfides of arsenic, antimony, and bismuth are likely to cause much trouble. If the sulfides are partly oxidized, however, enough acids and soluble sulfates may be formed to give serious trouble in the cyanide process. Pyrrhotite (Fe_nS_{n+1}) is slowly dissolved by cyanide solutions, consuming cyanide, and also has the power of absorbing the oxygen that enters into the reaction.

Base-metal sulfides are seldom abundant enough in gold ores to make it profitable to save and market them separately. Occasionally, however, their presence, even in small amounts, has caused the treatment to be modified slightly where they can be recovered cheaply to furnish an additional source of revenue. At the Alaska-Juneau mine lead is present in very small quantities (a fraction of 1 percent) but the tonnage treated is large (currently about 11,000 tons per day) and the aggregate amount of lead is worth saving in the form of a concentrate, which is shipped for smelting.

Arsenic and antimony if present in appreciable quantities cause serious trouble. Sulfarsenides and sulfantimonides are present in many gold ores, and these, together with the sulfides, cause "sickening" of the mercury in the amalgamation process, chiefly by coating it with a black film that causes the mercury to separate into small globules. Partly decomposed arsenic and antimony minerals and the oxides realgar and orpiment are particularly injurious. Some arsenic and antimony compounds are soluble in cyanide solutions and cause high consumption of cyanide; they also tend to foul the solutions to such an extent that gold and silver cannot be dissolved, or they may form reducing compounds that will precipitate gold already dissolved.

Tellurides of gold and silver, important ore minerals in some districts, do not amalgamate directly and cause sickening of the mercury as well. Sodium amalgam will amalgamate with them, however. It long has been thought that they are not amenable to ordinary cyanidation but are soluble in bromocyanide, and the usual practice has been to roast telluride ores before cyanidation. Recent research has demonstrated that tellurides are soluble in cyanide if ground fine enough, but it is probable that the extra expense of fine-grinding often will exceed the cost of preliminary roasting to obtain equally good extraction.

Certain metallic oxides, carbonates, hydrates, sulfates, and arsenates, as well as soluble sulfides, form a group known collectively as cyanicides. They react with cyanide, forming double cyanides, which dissolve gold and silver very slowly, if at all, or they may decompose the cyanide and thus destroy its value as a solvent. Soluble copper salts are perhaps the most troublesome cyanicides, for, in addition to reacting with the cyanide, they deoxidize the solutions. Along with ferrous sulfide, sulfate, and other salts, their effect cannot be counteracted by the addition of lime to the pulp but they can be partly removed by a preliminary acid or ammonia wash. Partial oxidation of sulfide minerals may prevent successful direct concentration by flotation, although special conditioning reagents may be used to make them amenable to this process.

The principal gold-milling methods include hand-sorting, amalgamation, gravity concentration, flotation concentration, cyanidation, and various combinations of two or more of these methods; chlorination is seldom employed in modern plants, though at one time it was used to a considerable extent.

In all of these methods except hand sorting, crushing and grinding of the crude ore are the preliminary steps, and in some instances where coarse crushing is done in stages, sorting may follow the first stage.

AMALGAMATION

There are two principal methods of amalgamation—plate amalgamation and barrel amalgamation.

In plate amalgamation the ore is crushed wet in stamp mills or ground in ball mills and the resulting pulp flows over copper plates (often silvered), which are coated with amalgam and mercury. Particles of free gold and silver coming in contact with the mercury alloy with it to form gold-silver amalgam, which adheres to the plates. Where stamps are employed, the screen openings range from about 12 to 40 mesh as a rule, which determines the maximum size of ore particles in the pulp passing over the plates.

Mercury sometimes is introduced into the stamp mortar, also, especially if there is much coarse free gold in the ore. The plates are dressed every few hours, the interval depending on the grade of the ore, amount of sickening that takes place, the scouring action of the pulp, and degree of discoloration of the plates by chemical action. Dressing usually comprises brushing the plates from the bottom upward with a whiskbroom or rubber scraper to remove excess amalgam, sprinkling fresh mercury on the resulting bare or thin spots, and working the mercury until the surface shows the proper amalgam condition. At longer intervals the plates are thoroughly cleaned and re-dressed with mercury. The gold is finally recovered by squeezing out excess mercury, retorting the amalgam, melting down the resulting sponge gold and casting it into bars.

The stamp mill is not an efficient crushing machine, although it has been retained in a number of mills, some of them quite large, probably owing to the fact that the investment had been made or to the peculiar crushing characteristics of the ores. It has some advantages in mills of small capacity because of the small unit capacity of each battery, which permits shutting down one or more batteries without lessening the efficiency of the others, and because of the comparatively low first cost.

Figure 143 is the flow sheet of the Argonaut mill as it was in 1928, which was typical of stamp-mill plate-amalgamation practice in the Mother Lode (California) district at that time. After passing over the plates, the pulp went to vanners for gravity concentration, and the vanner tails were cyanided in a separate plant. Figure 144 is the flow sheet of the south mill, Homestake mine, in 1927, which employed two-stage grinding and amalgamation; crushing is done by stamps and grinding by rod, tube, and ball mills.

Barrel amalgamation usually is employed for treatment of heavy concentrates from corduroy or sand tables or from coarse-gold traps and other small batches of rich material. The barrel is a revolving steel cylinder. It is charged with a batch of concentrate and a light load of grinding balls to break up the material and to scour and brighten the surfaces of the gold particles. After the barrel has rotated for several hours, an excess of mercury is added to it and it is revolved for 2 or 3 hours more. The batch is then discharged, the mercury and amalgam are recovered by panning, and the gold is recovered from the amalgam by retorting. Pan amalgamation is employed similarly for treating small batches of very rich concentrate; the concentrate is ground with mercury in cast-iron tubs or pans equipped with rotating dies or grinding shoes.

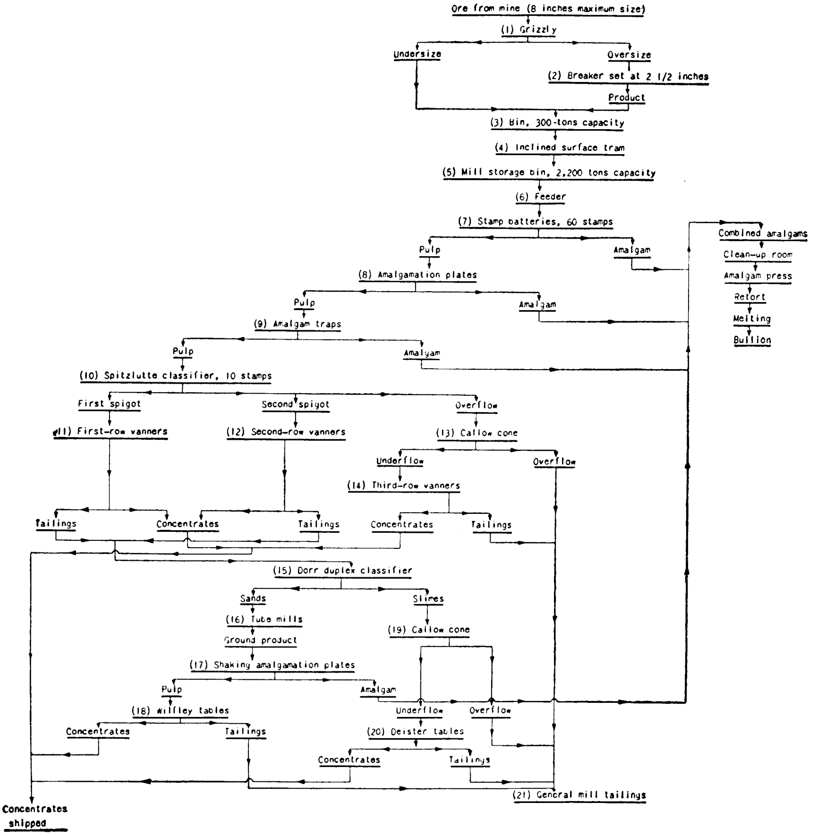
Ore from mine (8 inches maximum size)
(1) Grizzly
Undersize
Oversize
(2) Breaker set at 2 1/2 inches
Product
(3) Bin, 300-tons capacity
(4) Inclined surface tram
(5) Mill storage bin, 2,200 tons capacity
(6) Feeder
(7) Stamp batteries, 60 stamps
Combined amalgams
Clean-up room
Amalgam press
Retort
Melting
Bullion
Pulp
Amalgam
(8) Amalgamation plates
Pulp
Amalgam
(9) Amalgam traps
Pulp
Amalgam
(10) Spitzlutte classifier, 10 stamps
First spigot
Second spigot
Overflow
(11) First-row vanners
(12) Second-row vanners
(13) Callow cone
Underflow
Overflow
(14) Third-row vanners
Tailings
Concentrates
Tailings
Concentrates
Tailings
(15) Dorr duplex classifier
Sands
Slimes
(16) Tube mills
Ground product
(19) Callow cone
(17) Shaking amalgamation plates
Pulp
Amalgam
(18) Wilfley tables
Underflow
Overflow
Concentrates
Tailings
(20) Deister tables
Concentrates
Tailings
(21) General mill tailings
Concentrates shipped

FIGURE 143.—Flow sheet of Argonaut mill, California.

CONCENTRATION

Concentration is employed for the purpose of segregating the gold in a product of relatively small bulk for direct shipment to a smelter or for further treatment in the mill. For many years gravity concentration was virtually the only concentrating method employed in gold mills, but in recent years flotation methods have been used widely, especially in the flow sheets of new mills.

As an adjunct to other methods, concentration may effect marked savings in operating costs. Thus, it is often possible to produce a concentrate containing most of the gold after relatively coarse grinding, then fine-grind the concentrate preparatory to extracting the gold by cyanidation or amalgamation. Since fine-grinding usually is the most costly operation in gold milling, it is obvious that where it can be restricted to a concentrate, the bulk of which is only a fraction (sometimes a very small fraction) of that of the original ore, the saving in cost is appreciable. Furthermore, the constituents of the ore that are most abrasive and difficult to grind are often the gangue minerals that are largely removed by concentration.

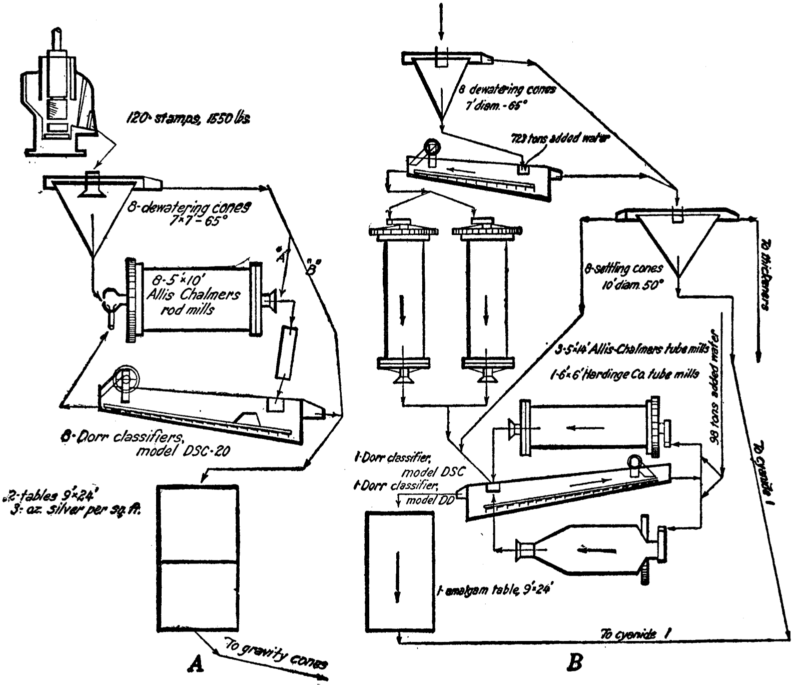
120 stamps, 1550 lbs.
8-dewatering cones 7×7 – 65°
"A"
"B"
8-5'×10' Allis Chalmers rod mills
8-Dorr classifiers, model DSC-20
2-tables 9'×24' 3 oz. silver per sq. ft.
To gravity cones
A
8 dewatering cones 7' diam. - 65°
723 tons added water
8-settling cones 10' diam. 50°
To thickeners
3-5'×14' Allis-Chalmers tube mills
1-6'×6' Hardinge Co. tube mills
98 tons added water
To cyanide 1
1-Dorr classifier, model DSC
1-Dorr classifier, model DD
1-amalgam table, 9'×24'
To cyanide 1
B

FIGURE 144.—Flow sheet of south mill, Homestake mine, South Dakota. *A*, Coarse-grinding section, 1927. *B*, Fine-grinding section, 1927.

The principal gravity-concentrating devices now employed are stationary blanket tables, concentrating cones, jigs, shaking tables, and vanners. Concentrating cones or traps frequently are employed to remove coarse gold from the circuit before it passes to the main treatment process; they are commonly placed between the primary ball or rod mill and a classifier operating in closed circuit therewith. Jigs are installed for a similar purpose at some point near the head of the flow sheet.

Blanket tables are stationary sloping surfaces covered with corduroy, coco matting, or other coarse-woven fabric. Corduroy especially manufactured for this purpose is probably the best material to use because of its durability and the riffling action of the ribs of the cloth. The pulp from the grinding mills is passed in a thin stream over the table, and the coarse gold and heavier minerals are caught and held on the cloth. The blankets are taken from the tables and washed in a tub or vat at frequent intervals to remove the concentrate, which is then treated in an amalgam barrel, cyanided, or shipped to a smelter.

Shaking tables are of several types, but all operate on the same basic principle. They consist of a flat, sloping deck, fitted with thin, tapered riffles running parallel or diagonally with the long dimension of the deck (for sand feed) or are covered with fabric (for slime feed). An eccentric mechanism is employed to oscillate the deck in a longitudinal direction. The pulp, which should be sized or classified for best results, is fed in a thin stream over the deck, the light gangue minerals being washed over the lower side of the table whereas the gold and heavy minerals are caught behind the riffles and moved along toward the end of the table by the jerking motion.

Figure 145 is a flow sheet of the Alaska-Juneau mill (1929), in which tables are employed to produce a high-grade gold concentrate that is treated by amalgamation, a middling or shipping concentrate, and a tailing product.

Vanners formerly were employed extensively for concentration of slime material, but a low extraction usually resulted because of the tendency of very fine gold particles to float off with the gangue. Usually, appreciably higher extraction of gold from slimes can be obtained by flotation or cyanidation.

Classification is a form of concentration that usually is employed more specifically for the purpose of separating the coarse from the fine material (the sands from the slimes) and in the usual gold-mill flow sheet is employed in the grinding section to separate coarse material that requires regrinding from the fine slimes that are ready for the next stage of the process. Classifiers may be either in closed circuit with a grinding mill (usually a ball, pebble, or rod mill), in which case the sands are returned to the mill from which they were received, or in series with a second mill.

Flotation is used principally on gold ores that carry enough sulfide mineral to stabilize a froth, although it has been applied successfully in connection with other methods to the treatment of oxidized ores containing little if any sulfide mineral. The flotation concentrate may be a finished product that is shipped to a smelter or it may be treated further for extraction of the gold by direct cyanidation or by roasting followed by cyanidation. Flotation reagents are now available that will not inhibit amalgamation, and flotation concentrates are sometimes treated by amalgamation for recovery of the gold.

In some mills a quick recovery of coarse, rich material is accomplished in a "unit cell" placed in the grinding section, usually between the ball mill and classifier or after the classifier and just ahead of the main flotation circuit, which thus performs a function similar to that of a concentrating cone or jig placed at the same point in the circuit as mentioned previously.

Figure 146 is the flow sheet of a small mill at Burgdorf, Idaho, in which a jig follows the ball mill and the concentrate from the jig is cleaned on a Wilfley table and amalgamated in a barrel. The jig tailing and the Wilfley table tailing are classified, the sands being returned to the ball mill for regrinding while the slime ("overflow" in figure) is concentrated on tables covered with rubber matting. The tailing from the tables is treated in a five-cell flotation machine. In this mill, therefore, flotation is employed as an adjunct to gravity concentration and amalgamation to recover additional gold from the tailing.

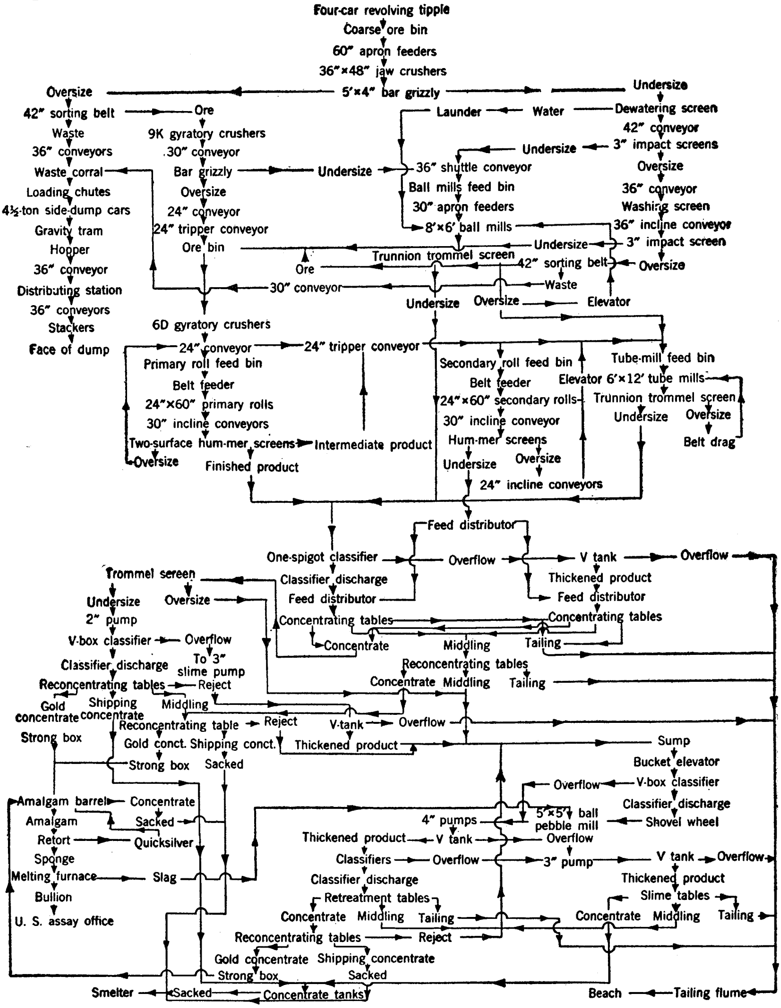
Four-car revolving tipple
Coarse ore bin
60" apron feeders
36"×48" jaw crushers
5'×4" bar grizzly
Oversize
Undersize
42" sorting belt
Ore
Waste
9K gyratory crushers
36" conveyors
30" conveyor
Waste corral
Bar grizzly
Undersize
Loading chutes
Oversize
4½-ton side-dump cars
24" conveyor
Gravity tram
24" tripper conveyor
Hopper
Ore bin
36" conveyor
Distributing station
36" conveyors
Stackers
Face of dump
Launder
Water
Dewatering screen
42" conveyor
Undersize
3" impact screens
36" shuttle conveyor
Oversize
Ball mills feed bin
36" conveyor
30" apron feeders
Washing screen
8'×6' ball mills
36" incline conveyor
Undersize
3" impact screen
Trunnion trommel screen
Ore
42" sorting belt
Oversize
30" conveyor
Waste
Undersize
Oversize
Elevator
6D gyratory crushers
24" conveyor
24" tripper conveyor
Primary roll feed bin
Belt feeder
24"×60" primary rolls
30" incline conveyors
Two-surface hum-mer screens
Intermediate product
Oversize
Finished product
Secondary roll feed bin
Belt feeder
24"×60" secondary rolls
30" incline conveyor
Hum-mer screens
Undersize
Oversize
24" incline conveyors
Elevator
Tube-mill feed bin
6'×12' tube mills
Trunnion trommel screen
Undersize
Oversize
Belt drag
Feed distributor
One-spigot classifier
Overflow
V tank
Overflow
Classifier discharge
Thickened product
Feed distributor
Feed distributor
Concentrating tables
Concentrating tables
Concentrate
Middling
Tailing
Reconcentrating tables
Concentrate
Middling
Tailing
Trommel screen
Undersize
Oversize
2" pump
V-box classifier
Overflow
To 3" slime pump
Classifier discharge
Reconcentrating tables
Reject
Gold concentrate
Shipping concentrate
Middling
Strong box
Reconcentrating table
Reject
V-tank
Overflow
Gold conct.
Shipping conct.
Thickened product
Strong box
Sacked
Sump
Bucket elevator
Overflow
V-box classifier
Classifier discharge
Amalgam barrel
Concentrate
Amalgam
Sacked
5'×5' ball pebble mill
Shovel wheel
Retort
Quicksilver
4" pumps
Thickened product
V tank
Overflow
Sponge
Classifiers
Overflow
3" pump
V tank
Overflow
Melting furnace
Slag
Classifier discharge
Thickened product
Bullion
Retreatment tables
Slime tables
U. S. assay office
Concentrate
Middling
Tailing
Concentrate
Middling
Tailing
Reconcentrating tables
Reject
Gold concentrate
Shipping concentrate
Strong box
Sacked
Smelter
Sacked
Concentrate tanks
Beach
Tailing flume

FIGURE 145.—Flow sheet of Alaska-Juneau mill, Alaska.

Figure 147 is the flow sheet of a small gold concentrator in which straight flotation is employed, the overflow from the classifiers being fed directly to two sets of flotation cells in parallel, the cleaner cells of which produce a finished concentrate for shipment to the smelter.

There are so many different ways in which flotation is combined with other methods of recovery that neither figure 146 nor figure 147 can be said to be typical; they merely represent two extremes, one in which flotation is an adjunct to the mill for recovering gold in the mill the other in which flotation is the only process. Flotation concentrates often are treated by cyanidation, usually after regrinding, in which case reagent strength and consumption per ton usually are higher than when raw ores are cyanided.

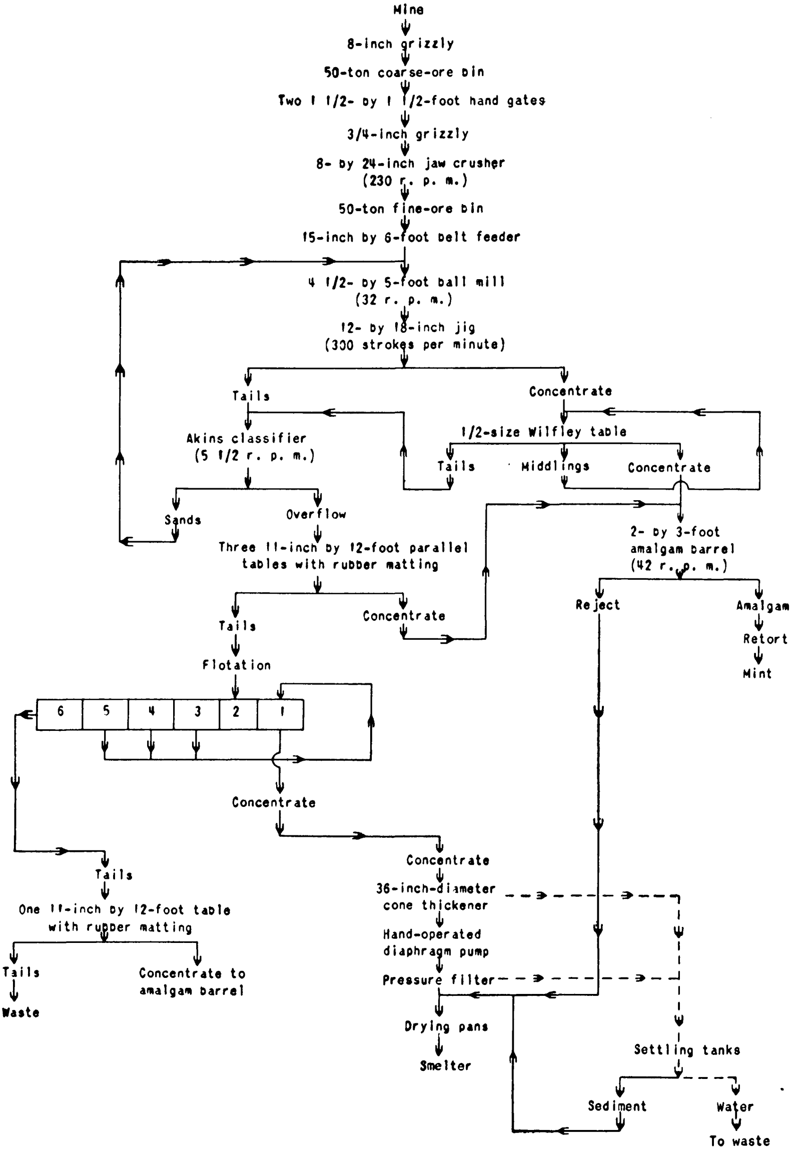
Mine
8-inch grizzly
50-ton coarse-ore bin
Two 1 1/2- by 1 1/2-foot hand gates
3/4-inch grizzly
8- by 24-inch jaw crusher (230 r. p. m.)
50-ton fine-ore bin
15-inch by 6-foot belt feeder
4 1/2- by 5-foot ball mill (32 r. p. m.)
12- by 18-inch jig (300 strokes per minute)
Tails
Concentrate
Akins classifier (5 1/2 r. p. m.)
1/2-size Wilfley table
Tails
Middlings
Concentrate
Sands
Overflow
Three 11-inch by 12-foot parallel tables with rubber matting
2- by 3-foot amalgam barrel (42 r. p. m.)
Reject
Amalgam
Retort
Mint
Tails
Concentrate
Flotation
6
5
4
3
2
1
Concentrate
Concentrate
Tails
36-inch-diameter cone thickener
One 11-inch by 12-foot table with rubber matting
Hand-operated diaphragm pump
Tails
Waste
Concentrate to amalgam barrel
Pressure filter
Drying pans
Smelter
Settling tanks
Sediment
Water
To waste

FIGURE 146.—Flow sheet of Golden Anchor mill, Burgdorf, Idaho.

CYANIDATION

The cyanide process is based on the work of MacArthur and the Forrests (begun in 1886) and involves the leaching of gold and silver from alkaline pulps with dilute solutions of sodium or potassium cyanide, the filtering and clarification of the gold-silver-bearing solution, and precipitation of metals from solution. Oxygen is essential to the solution reaction, and in practice oxygenation is accomplished by violent agitation of the pulp by the injection of compressed air. Precipitation usually is by means of zinc shavings or zinc dust and is improved if the solution be first de-aerated. Aluminum is used instead of zinc under some unusual conditions.

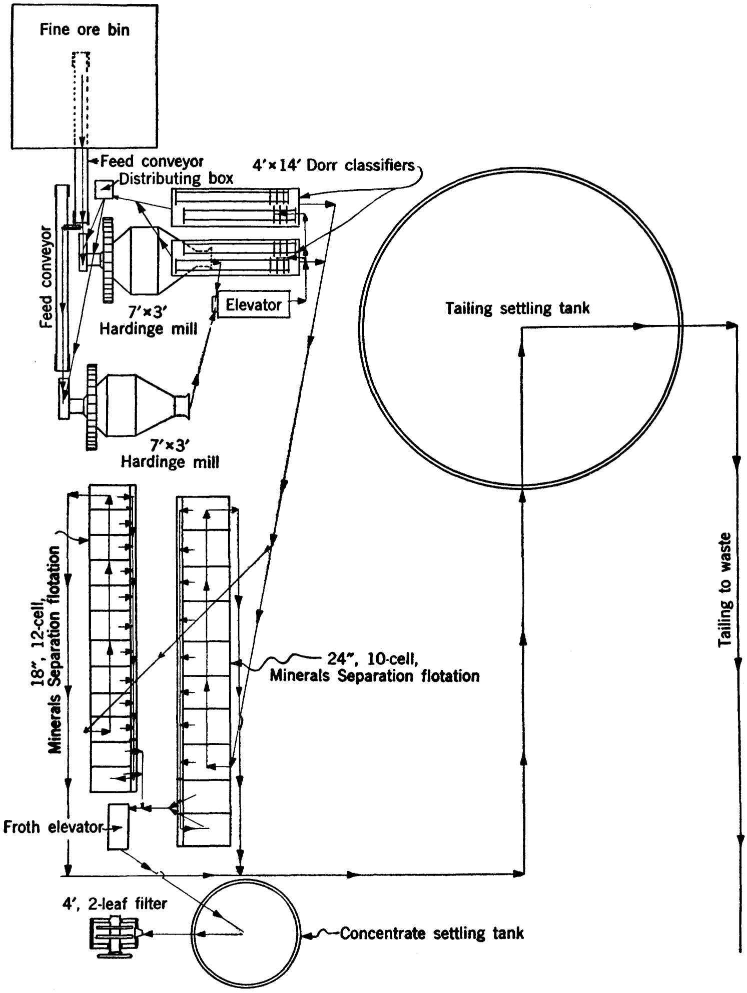
Fine ore bin
Feed conveyor
Distributing box
4′×14′ Dorr classifiers
Feed conveyor
7′×3′ Hardinge mill
Elevator
7′×3′ Hardinge mill
Tailing settling tank
18″, 12-cell, Minerals Separation flotation
24″, 10-cell, Minerals Separation flotation
Tailing to waste
Froth elevator
4′, 2-leaf filter
Concentrate settling tank

FIGURE 147.—Flow sheet of small gold concentrator using straight flotation.

Nearly all gold ores are amenable to cyanidation, although, as previously pointed out, certain constituents of some ores cause excessive consumption of cyanide and other reagents, or the fouling of solutions, with resultant high costs. A few are so difficult to treat by direct cyanidation that costly preliminary treatment must be employed, when other methods might prove more economical. The principal cyanicides have been mentioned already. Carbonaceous matter, often contained in graphitic slates associated with gold ores, may result in high tailing loss through its capacity to precipitate gold from the solutions.

In general, cyanide treatment comprises four steps—preparation of the ore (crushing, screening, grinding, and classification), leaching or dissolving the gold, separation of the gold-bearing ("pregnant") solution from the leached pulp (thickening, filtration, and clarification of the solution), and precipitation.

Preparation of the ore as above defined is very important, and as it is usually the most expensive part of the process (especially fine grinding), it has been the object of much research and experimentation. Gow, Guggenheim, and Coghill reviewed the subject of fine grinding in ore concentrators in 1933.[85]

Data on equipment, fineness of grinding, costs, etc., at gold mills will be tabulated with other gold-milling data later.

Where an all-cyanide method is employed, the grinding usually is done in cyanide solution and no water is used in the mill except for final washing of the filter cake. A good deal of the gold, sometimes most of it, is then dissolved in the grinding section of the mill and the time of contact of solution with pulp in agitators and thickeners may be reduced appreciably.

Leaching is accomplished in different ways, according to the character of the ore; the entire mill feed may be finely ground and treated by the agitation or all-slime process, it may be separated by classification into sand and slime portions that are cyanided separately or concentrate may be cyanided by one method and tailing by another. There are two principal leaching methods—batch or "sand" leaching and all-slime or agitation leaching.

Sand leaching is done in tanks fitted with fiber bottoms, into which the sands are charged. Successive leachings with cyanide solution are made, each usually being weaker than the preceding one. The solutions are either introduced at the top and percolate down through the pulp, or enter at the bottom under a slight pressure and are decanted after passing upward through the pulp. Draining and leaching in stages usually gives better results than continuous passage of the solution, since the charge is better aerated in this way. For successful leaching, the charge should be of classified material and free from slimes that would clog the pulp column, preventing passage of the solution in places and causing "channeling" in others. Thus, in large plants where sand leaching is practiced, it is customary to treat only the sands by this process, the slimes being treated by agitation and countercurrent decantation or (and) filter washing. The sand charge is given one or more final water washes and is discharged from the tank.

All-slime or agitation leaching depends on constant movement of the ore particles in well-aerated cyanide solution. Agitation and solution usually begin in the grinding circuit, as previously stated. After grinding and classification of the pulp to the desired degree of fineness, it is usually first thickened and then fed to agitation tanks, where it is brought to the required consistency and cyanide strength by addition of stock cyanide solution. Agitation is accomplished by mechanical means, by compressed air, or both. The time of agitation required to obtain extraction of the gold varies and must be determined in each instance by tests and experience. Agitation in batches, while it can be controlled rather closely, is usually more costly than the continuous process that is used almost universally in modern plants.

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[85] Gow, Alexander M., Guggenheim, Morris, and Coghill, Will H., Review of Fine Grinding in Ore Concentrators: Bureau of Mines Inf. Circ. 6757, 1934, 29 pp.

In the agitation processes the pulp from the primary thickener goes to agitation tanks, of which there are usually at least three in series. After leaving the last agitator, the pulp may be thickened and washed one or more times and finally filtered, or it may be subjected to continuous countercurrent decantation. Most of the cyanide plants in Ontario and Quebec employ filter washing rather than countercurrent decantation. If grinding has been done in cyanide solution, the overflow from the primary thickener preceding the agitators usually constitutes the gold solution that goes to the clarification and precipitation section of the mill.

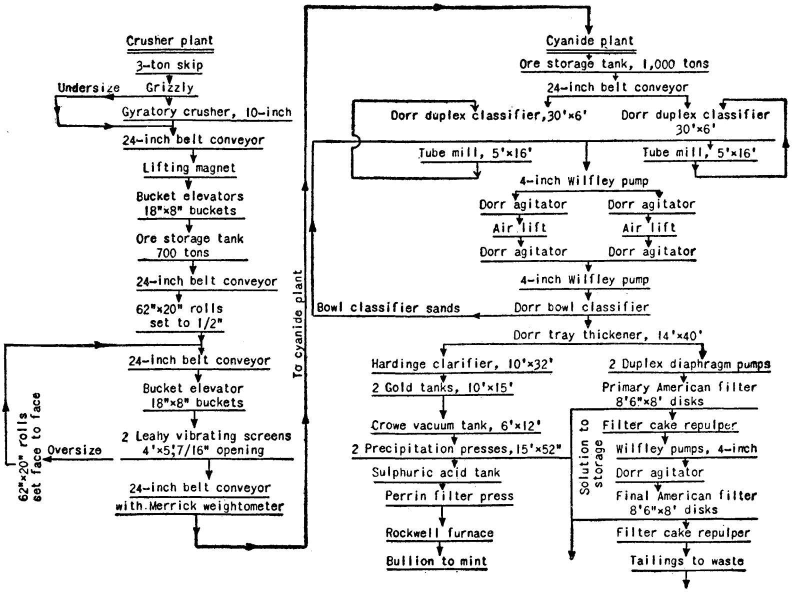
Crusher plant
3-ton skip
Undersize
Grizzly
Gyratory crusher, 10-inch
24-inch belt conveyor
Lifting magnet
Bucket elevators 18"×8" buckets
Ore storage tank 700 tons
24-inch belt conveyor
62"×20" rolls set to 1/2"
24-inch belt conveyor
Bucket elevator 18"×8" buckets
2 Leahy vibrating screens 4'×5', 7/16" opening
Oversize
62"×20" rolls set face to face
24-inch belt conveyor with Merrick weightometer
To cyanide plant
Cyanide plant
Ore storage tank, 1,000 tons
24-inch belt conveyor
Dorr duplex classifier, 30'×6'
Dorr duplex classifier 30'×6'
Tube mill, 5'×16'
Tube mill, 5'×16'
4-inch Wilfley pump
Dorr agitator
Dorr agitator
Air lift
Air lift
Dorr agitator
Dorr agitator
4-inch Wilfley pump
Bowl classifier sands
Dorr bowl classifier
Dorr tray thickener, 14'×40'
Hardinge clarifier, 10'×32'
2 Gold tanks, 10'×15'
Crowe vacuum tank, 6'×12'
2 Precipitation presses, 15'×52"
Sulphuric acid tank
Perrin filter press
Rockwell furnace
Bullion to mint
2 Duplex diaphragm pumps
Primary American filter 8'6"×8' disks
Solution to storage
Filter cake repulper
Wilfley pumps, 4-inch
Dorr agitator
Final American filter 8'6"×8' disks
Filter cake repulper
Tailings to waste

FIGURE 148.—Flow sheet of Coniaurum mill, Ontario.

In straight filtration-leaching, the process is briefly and essentially as follows: The pulp from the agitators may be thickened first or may be sent directly to the continuous filters. On the filters the cake first receives one or more washes with barren cyanide solution and then one or more of water, both filtrates being pumped back to the mill-solution tank for use in the grinding and other sections of the mill. Figure 148 is a flow sheet of the Coniaurum mill (1931) in which filtering, repulping and agitation, and secondary filtering are employed.

In countercurrent decantation the pulp from the agitators goes to three or more thickeners arranged in series. These thickeners are often of the classifier-thickener type. The leached pulp enters the first thickener from the last agitator (if there is no primary thickener preceding the agitators, this may be considered the primary thickener). Most of the pregnant solution overflows this thickener and either goes back to the grinding circuit or directly to clarification and precipitation, while the thickened underflow is discharged to the second thickener of the series. In this thickener the pulp is mixed with much lower grade overflow from the third tank. After mixing in the second tank, most of the dissolved gold that came in with the pulp is dispersed throughout the solution, raising its value. The overflow from the second tank goes, in turn, to the first tank, while the thickened pulp is discharged to the third tank of the series, carrying with it some solution of lower grade than that which it brought with it from the first thickener. The process is repeated in each tank, the pulp becoming successively leaner as regards the value of the solution carried along with it whereas the main body of solution is enriched at each step. At the last thickener in the series enough fresh water or weak barren solution is added to offset the amount of liquid leaving the circuit with the leached slime plus spillage, evaporation, and other losses, and the slime goes to the tailing pond or to filtration for further washing. Countercurrent decantation is illustrated by the simple flow sheet of the Big Jim mill (fig. 149).

The subject of gold milling was discussed in a general way but in somewhat more detail in an earlier bulletin.[86] Progress reports covering results of research on gold milling are issued from time to time by the Metallurgical Division of the Bureau of Mines.[87]

GOLD-MILLING DATA

The data in table 57 have been obtained from various sources as noted. In view of the frequent changes in practice that take place in most mills due to changes in ore characteristics, in methods and equipment brought about by research and improved machines, and increases or decreases in capacity, these data should be understood to apply specifically to the period covered and only generally to long-time operations. However, they represent broadly current practices and results obtained in milling various types of gold ores.

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[86] Jackson, Charles F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, pt. 3, pp. 89–138.

[87] Leaver, E. S., Woolf, J. A., and Head, R. E., Progress Report, Metallurgical Division. 2. Gold Recovery Studies: Bureau of Mines Rept. of Investigations 3226, 1934, 31 pp.

Leaver, Edmund S., Boyer, M. B., Woolf, J. A., Head, R. E., Gandrud, B. W., Evans, R. E., and Thackwell, F. W., Progress Report, Metallurgical Division. 11. Studies on the Recovery of Gold and Silver: Bureau of Mines Rept. of Investigations 3275, 1935, 65 pp.

Staff of the Ore-Testing Section, Progress Reports, Metallurgical Division. 22. Ore-Testing Studies, 1936–37: Bureau of Mines Rept. of Investigations 3370, 1938, 174 pp.

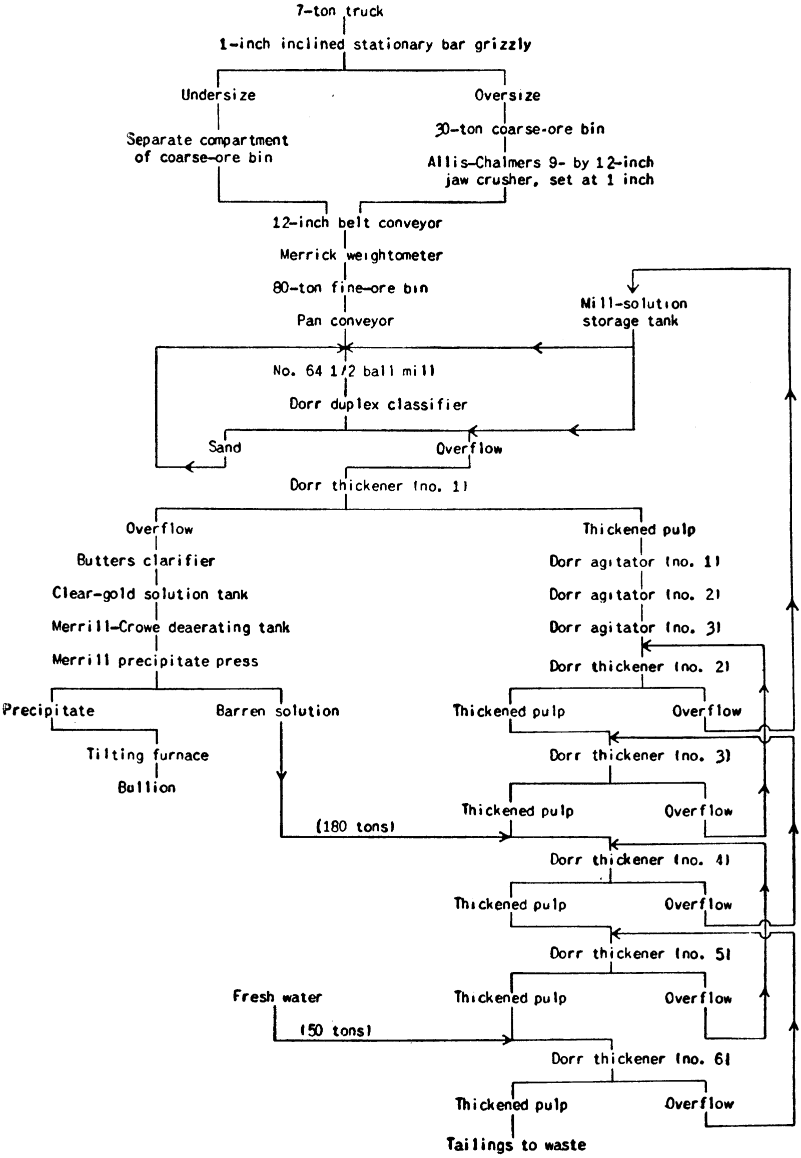
7-ton truck
1-inch inclined stationary bar grizzly
Undersize
Oversize
30-ton coarse-ore bin
Separate compartment of coarse-ore bin
Allis-Chalmers 9- by 12-inch jaw crusher, set at 1 inch
12-inch belt conveyor
Merrick weightometer
80-ton fine-ore bin
Mill-solution storage tank
Pan conveyor
No. 64 1/2 ball mill
Dorr duplex classifier
Sand
Overflow
Dorr thickener (no. 1)
Overflow
Thickened pulp
Butters clarifier
Dorr agitator (no. 1)
Clear-gold solution tank
Dorr agitator (no. 2)
Merrill-Crowe deaerating tank
Dorr agitator (no. 3)
Merrill precipitate press
Dorr thickener (no. 2)
Precipitate
Barren solution
Thickened pulp
Overflow
Tilting furnace
Dorr thickener (no. 3)
Bullion
Thickened pulp
Overflow
(180 tons)
Dorr thickener (no. 4)
Thickened pulp
Overflow
Dorr thickener (no. 5)
Fresh water
Thickened pulp
Overflow
(50 tons)
Dorr thickener (no. 6)
Thickened pulp
Overflow
Tailings to waste

FIGURE 149.—Flow sheet of Big Jim mill, Arizona.

TABLE 57.—*Gold-milling data*

1. GENERAL DATA

Mine, reference, and year	Nature of the ore treated		Gold		General method of treatment
	Country rock	Associated minerals and rocks	Occurs	Heads, assay value	
Porcupine-United, Ontario, 1930–31.[1]	Basaltic schist	Quartz, pyrite	75 percent free; balance in sulfides.	$11.00 [2]	Amalgamation and gravity concentration. Gold and amalgam caught in sumps, traps, and blankets. Shaking-table concentrates cyanided in another plant.
Argonaut, Calif., 1929 [3]	Slate and greenstone	Quartz, arsenopyrite, sphalerite, galena, chalcopyrite, pyrrhotite, tetrahedrite.	81 percent free	$6.07 [2]	Stamp milling, plate amalgamation, concentration. Tailing cyanided in separate plant.
Alaska-Juneau, Alaska, 1929 [4]	Slate and metagabbro.	Quartz, some pyrite, pyrrhotite, galena, sphalerite.	Mostly free	$1.100 [2] [5] $1.919 [2] [6]	Sorting to remove about 50 percent, gravity concentration, amalgamation.
Atlanta, Idaho, April 1934 [7]	Granite	Quartz, pyrite, arsenopyrite, silver.	Free and in sulfides	$16.34 [8]	Amalgamation followed by flotation.
Talache, Idaho, 1936 [9]	Rhyolite porphyry	Quartz, sericite, pyrite, sphalerite, galena, tetrahedrite, bismutite.	Mostly free	$7.00 [8]	Amalgamation (hydraulic traps, plates, amalgamation pans).
Pilgrim, Ariz., 1936 [10]	Rhyolite and tuffs, breccia.	Quartz, galena, pyrite, sphalerite, chalcopyrite, tetrahedrite.	Free	$9.25 [8] (gold only)	Flotation; amalgamation of flotation concentrate; amalgamation tailing to smelter (later amalgamation tailing was cyanided).
Cardinal, Calif., 1937 [11]	Quartzite	Quartz, pyrrhotite, arsenopyrite, pyrite, chalcopyrite, sphalerite.	Mostly in sulfides	$9.45 [8]	Gravity and flotation concentration.
Spring Hill, Mont., 1929–30 [12]	Diorite	Pyrite, arsenopyrite, diorite gangue.	10–15 percent free; balance in sulfides.	$6.46 [2]	Straight flotation concentration.
Premier, British Columbia, 1931.[13]		Quartz, pyrite, sphalerite, galena, copper sulfides.	With sulfides	$6.30 [2] (5.93 ounces silver).	Stage grinding; flotation; classification.
Golden Belt, Ariz., January 1936.[14]	Schist	Quartz, pyrite, galena	do	$7.35 [8] (gold only)	Hand sorting to remove one-third, followed by flotation.
Golden Turkey, Ariz., 1935 [14]	do	do	do	$5.59 [6] [8] (gold only)	Do.
Mammoth, Mont., 1935 [15]	Gneiss	Quartz, pyrite, chalcopyrite, galena, sphalerite.	do	$9.10 [8]	Straight flotation; concentrates cleaned on shaking tables.
Boss-Tweed, Mont., 1935 [15]	do	Quartz and pyrite	Free and with pyrites	$4.86 [8]	Straight flotation.
Mogollon Consolidated, N. Mex., 1936.[16]	Andesite breccia	Quartz, calcite, pyrite, argentite, sphalerite, galena.		$8.25 [6] [8] (gold and silver)	Sorting followed by flotation.
Peach, Ariz., 1936 [17]		Old stamp-mill tailings		$3.15 [8]	Straight flotation.
Golden Anchor, Idaho, 1937 [18]	Schist, gneiss, quartzite.	Quartz, oxidized minerals, pyrite, tetrahedrite, galena, sphalerite, molybdenite.	Mostly free	$24.88 [8] gold; 2.58 ounces silver.	Gravity concentration of sands and amalgamation. Flotation concentration of slimes and amalgamation.
Big Jim, Ariz., 1934 [19]	Andesite	Quartz and calcite; no sulfides.	Very fine		Cyanidation (grinding in solution, agitation and countercurrent decantation).
Kirkland Lake Gold, Ontario, 1930.[20]	Seyenite, porphyry, lamprophyre.	Silicified country rock; some tellurides and pyrite.	do	$11.38 [2]	Cyanidation (fine grinding in solution, agitation and countercurrent decantation, and filtration).
Coniaurum, Ontario, 1930 [21]	Schist	Quartz, mineralized schist, pyrite.	Free and in pyrite	$6.00 [2]	Cyanidation (grinding in solution, agitation, thickening, and filtration).

Yellow Aster Mine, Calif., 1936.[22]	Granite and schist	Oxidized ores; little pyrite, arsenopyrite, and scheelite.	Free		Crushing and screening: +1½-inch to waste, −1½ inch stamped and amalgamated.
Yellow Aster tailings		Tailing from present and old stamp mill.			Cyanidation (classifying, batch-leaching of sands; thickening, agitation, and filtering of slimes).
Tom Reed, Ariz., 1934–36 [23]	Andesite	Quartz, calcite, silicified andesite, fluorspar, adularia, iron oxides.	Very fine	$12.19 [8] gold $0.52 silver	2-stage grinding and classification in solution. Cyanidation (agitation and countercurrent decantation).
United Eastern, Ariz., 1917–25.[24]	do	do	do	$19.87 [2]	Do.
Black Hawk, N. Mex., 1936 [16]				$9.00 to $12.00 [25]	2-stage grinding and classification in solution, cyanidation (agitation, countercurrent decantation, filtering).
Siscoe, Quebec, December 1936 to February 1937.[26]	Granodiorite, schist	Quartz, pyrite, pyrrhotite, chalcopyrite, and schist.	Mostly free	$17.52	Sorting (10 percent removed), grinding in solution, gravity concentration, amalgamation, and cyanidation of tailings.
Hollinger, Ontario, 1934 [27]	Schist and porphyry	Quartz, pyrite, mineralized schist.	Free and in pyrite	$7.68 [28]	Grinding in solution; table concentration; concentrates reground and cyanided; tails cyanided by countercurrent decantation and filtering in parallel.
Golden Cycle, Colo., 1929 [29]	(30)	(30)	(30)	$10.38 [2]	(30).

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[1] Vary, R. A., Amalgamation Practice at the Porcupine United Gold Mines (Ltd.), Timmins, Ontario: Bureau of Mines Inf. Circ. 6433, 1931, 5 pp.

[2] Gold $20.67 per ounce.

[3] Woodworth, S. E., Milling Methods and Costs at the Argonaut Mill, Jackson, Calif.: Bureau of Mines Inf. Circ. 6476, 1931, 12 pp.

[4] Bradley, P. R., Milling Practice at the Alaska-Juneau Concentrator: Bureau of Mines Inf. Circ. 6236, 1930, 16 pp.

[5] Crude ore.

[6] Sorted ore.

[7] Gardner, E. D., Milling Methods and Costs at the Concentrator of the St. Joseph Lead Co., Atlanta, Idaho: Bureau of Mines Inf. Circ. 6836, 1935, 10 pp.

[8] Gold $35.00 per ounce.

[9] Skidmore, Joe H., Gold Mining and Milling Methods and Costs at the Gold Hill Mine of Talache Mines, Inc., Quartzburg, Idaho: Bureau of Mines Inf. Circ. 6985, 1938, 15 pp.

[10] Hastings, Earle F., Mining and Milling Methods at the Pilgrim Mine, Chloride, Ariz.: Bureau of Mines Inf. Circ. 6945, 1937, 18 pp.

[11] Lenhart, Walter B., Milling Methods and Costs of the Cardinal Gold Mining Co., Bishop Creek, Calif.

[12] Grant, L. A., Milling Methods and Costs at the Spring Hill Concentrator of the Montana Mines Corporation, Spring Hill, Mont.: Bureau of Mines Inf. Circ. 6411, 1931, 8 pp.

[13] Pitt, D. L., Asselstine, W. J., and Coulter, D. L., Milling Methods and Costs at the Concentrator of the Premier Gold Mining Co., Ltd., Premier, British Columbia: Bureau of Mines Inf. Circ. 6742, 1933, 18 pp.

[14] Guiteras, Jos. R., Gold Mining and Milling in the Black Canyon Area, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6905, 1936, 51 pp.

[15] Lorain, S. H., Gold Lode Mining in the Tobacco Root Mountains, Madison County, Mont.: Bureau of Mines Inf. Circ. 6972, 1937, 74 pp.

[16] Metzger, O. H., Gold Mining in New Mexico: Bureau of Mines Inf. Circ. 6987, 1938, 71 pp.

[17] Metzger, O. H., Gold Mining and Milling in the Wickenburg Area, Maricopa and Yavapai Counties, Ariz.: Bureau of Mines Inf. Circ. 6991, 1938, 78 pp.

[18] Lorain, S. H., and Buford, W., Mining and Milling Methods and Costs of the Golden Anchor Mining Co., Burgdorf, Idaho: Bureau of Mines Inf. Circ. 7024, 1938, 15 pp.

[19] Johnson, C. H., Mining and Milling Methods at the Big Jim Mine, Oatman, Ariz.: Bureau of Mines Inf. Circ. 6824, 1935, 12 pp.

[20] Dixon, John, Milling Practice of the Kirkland Lake Gold Mines (Ltd.), Kirkland Lake, Ontario: Bureau of Mines Inf. Circ. 6508, 1931, 13 pp.

[21] Redington, John, Milling Methods and Costs of the Coniaurum Mines (Ltd.), Schumacher, Ontario: Bureau of Mines Inf. Circ. 6541, 1931, 5 pp.

[22] Cooper, Corwin L., Mining and Milling Methods and Costs at the Yellow Aster Mine, Randsburg, Calif.: Bureau of Mines Inf. Circ. 6900, 1936, 21 pp.

[23] Brough, Paris V., Milling Methods and Costs at the Mill of the Tom Reed Gold Mines Co., Oatman, Ariz.: Bureau of Mines Inf. Circ. 6975, 1937, 25 pp.

[24] North, W. O., Mill Operations at United Eastern During 1917 and 1918: Trans. Am. Inst. Min. and Met. Eng., vol. 63, 1920, p. 548. Bagley, E. M., Operations at United Eastern Mill: Eng. and Min. Jour., vol. 119, 1925, p. 436.

[25] Gold 40 percent, silver 60 percent of total value.

[26] Djingheuzian, L. E., A Review of Siscoe Milling Practice: Trans. Canadian Inst. Min. and Met., vol. 40, pp. 602–613.

[27] Longmore, E. Lynden, and Williams, Maurice E., Milling (at Hollinger): Canadian Min. Jour., September 1935, pp. 405–414.

[28] Gold at $32.16 per ounce.

[29] Harner, L. S., Milling Methods and Costs at the Golden Cycle Mill, Colorado Springs, Colo.: Bureau of Mines Inf. Circ. 6739, 1934, 18 pp.

[30] Sulfotelluride ores from Cripple Creek occurring in breccia and granite treated by roasting and cyanidation; many small lots from various districts treated by direct cyanidation; complex sulfide custom ores treated by selective flotation.

Table 57.—*Gold-milling data*—Continued

1. GENERAL DATA—continued

Mine, reference, and year	Nature of the ore treated		Gold		General method of treatment
	Country rock	Associated minerals and rocks	Occurs	Heads, assay value	
Lone Pine, Idaho, 1937[31]	Quartzite	Quartz, galena, pyrite, sphalerite.	Free and with sulfides	(?)	Amalgamation.
Orogrande-Frisco, Idaho, 1937.[31]	Silicified schist and pegmatite.	Oxidized minerals, quartz, pyrite.	do	About $1.50	Cyanidation; sands leached, slime agitated and washed by countercurrent decantation. A jig concentrate is taken from minus 1/16-inch screen product from fine ore bin, is cleaned on Wilfley tables, and shipped to smelter.

2. CRUSHING PLANT

Mine	Primary crushers				Secondary crushers				Capacity of crushing plant, tons per hour
	Number	Type	Size, inches	Size of product, inches	Number	Type	Size	Size of product, inch or mesh	
Porcupine-United, Ontario	1	Jaw	7 x 10	−1 x 1½	1	Rolls	10 x 18 in	75 percent −½ in	3
Argonaut, Calif	1	do	16 x 18	−2½	60	Stamps	1,285 lb	−0.0277	30+
Alaska-Juneau, Alaska	[32] 3	do	36 x 48	−8	1	Gyratory	21 x 76 in	−2 in	600+
Atlanta, Idaho	1	do	14 x 24	−1¼	1	Rolls	14 x 36 in	−4 mesh	20
Talache, Idaho	1	do	10 x 20	−1½					15
Pilgrim, Ariz	1	do	8 x 15	−1½	1	Gyratory	No. 14½	30 percent+4 mesh; 4 percent −200 mesh.	15
Mossback, Ariz	1	do	8 x 15						
Strawberry, Mont	1	do	9 x 16		10	Stamps	900 lb	24 mesh	12
Cardinal, Calif	1	do	10 x 20	−4	1	Gyratory	20 in	½–¾ in	
Spring Hill, Mont	1 1	do do	24 x 36 14 x 18	−5 −1½	1	Cone	3 ft	−0.53 in	20
Premier, British Columbia	1	do	30 x 18	−3	1	Gyratory	No. 6 McCully	18.2 percent + 1.0 in.; 31.1 percent −0.25 in.	42
Golden Belt, Ariz	1	do	7 x 10	−1 to −1½					5
Golden Turkey, Ariz	1	do	10 x 16	−⅜					10
Mammoth, Mont	1	do	12 x 18	−1 to −1½					
Boss-Tweed, Mont	1	do	9 x 16	−1½					10
Mogollon Consolidated, N. Mex	1	do	9 x 15	−2	1	Cone	2 ft	−⅜ in	
Golden Anchor, Idaho	1	do	8 x 24	−½					

Big Jim, Ariz.	1	do	9 x 12	−1					11-12
Kirkland Lake Gold, Ontario	1	do	12 x 20	−3	1	Gyratory		−1½	20
Coniaurum, Ontario	1	Gyratory	10	−1¾	1	Primary roll	62 x 20 in	−½	45
					1	Secondary roll	do	11.1 percent + 4 mesh; 11.6 percent −200 mesh.	
Yellow Aster, Calif	1	Jaw		−2¼	50	Stamps	1,080 lb	−30 mesh	7.9
Tom Reed, Ariz	1	Gyratory	12	−2					40-50
	1	Jaw	12 x 24	−2					
Black Hawk, N. Mex	1	do	12 x 24	−1½	1	Cone	3 ft	−3/16 in	
Siscoe, Quebec	1	do	15 x 20	−4½	1	do	do	−¾ in	50
Hollinger, Ontario, 1934	[33] 1	do	48 x 60	−6	[34] 1	Rolls	62 x 24 in	do	500
	3	Gyratory	No. 7½	−2½					
	4	do	No. 5	5 percent +1½.	[35] 3	Rolls	62 x 24 in	−⅜ in	
Golden Cycle, Colo	2	Jaw	13 x 24	−3½ to −4	2	do	40 x 14 in	7.7 percent + 2 in.; 0.1 percent −200 mesh.	100
					1	Cone	5½ ft	−½ in	
Lone Pine, Idaho	1	do	7 x 10		1	Jaw	8 x 12		
Orogrande-Frisco, Idaho	1	do	24 x 36		1	Gyratory	No. 5 Gates	−1 in	
					1	Rolls	16 x 36	−3/16 in	

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[31] Lorain, S. H., Gold Mining and Milling in Idaho County, Idaho: Bureau of Mines Inf. Circ. 7039, 1938, 90 pp.

[32] Only 2 in use at one time.

[33] Located underground.

[34] Primary.

[35] Secondary.

TABLE 57.—*Gold-milling data*—Continued

3. FINE-GRINDING PLANT

Mine	Grinding and classification							New feed per hour, tons	Skeleton-size analysis of—			
	Mills			Circuit	Grinding media		Circulating load, per cent		Feed		Product	
	Number	Kind	Size		Kind of balls	Size of balls, inches			Per cent	Inch or mesh	Per cent	Mesh classifier overflow
Porcupine-United, Ontario.	1	Ball mill	4½ ft. x 16 in.	Closed with plates, blankets, and classifier.	Forged steel	4	125	1.04	75	½-in.	65	−200
Alaska-Juneau, Alaska.	2	Gyratory	11 x 36 in		Used as stand-by			750				
	1	do	6 in									
	4	Cone	4 ft	Closed with screens and coarse rolls.				600	7.4 92.4	+5 +1.5 in	13.8 12.1	+2 in. −¼ in.
		Coarse rolls	24 x 60 in	Parallel, closed with screens.			100	90	20	+½ in	1.7	+10
		Fine rolls	do				200	40	13	−7 mesh	28.9	−150
	11	Ball mills	8 x 6 ft	Closed with screen and belt drags.	Forged chrome steel.	6	135–150	165	1.8 5.6	+2.5 in −7 mesh	4.3 27.5	+10 −200
Atlanta, Idaho	1	Ball mill	4 x 12 ft	Open	Forged steel	4 and 3		8	100	−½ in		
	1	do	6 x 6 ft	Closed with classifier and secondary amalgam plates.	do	2½	250				80	−200
Talache, Idaho	1	do	8 ft. x 22 in	Closed with trap and screen.	do	5		5.5		−1½ in		
	1	do	4 x 3 ft	Regrind mill	do	2		1.25				
Pilgrim, Ariz	1	do	6 ft. x 36 in	Closed with screen, unit cell, and classifier.	?	?		3.6	30 4	+4 mesh −200 mesh	2 37 53	+4 −200 sands −200 slime
Mossback, Ariz	1	do	4½ ft. x 16 in.	Closed with classifier	?	?	?	.8	?	?	95	−200
Cardinal, Calif	1	Rod mill	4 x 10 ft	Open	Steel rods	4 in. diam.		6.6			4.9 29.0	+28 −200
	1	Ball mill	5 x 6 ft	Closed with classifier and tables.	Pressed-steel balls.	4						
	1	do	7 x 6 ft	do	do	4		6.6				
Spring Hill, Mont	2	Ball mills	7 ft. x 36 in	Mills parallel closed with classifiers.	White cast-iron balls.	4	500	8.4 (2 mills)	6.7 5.6	9/16 x 1 in −100 +150	3.85 59.37	+80 −200

Premier, British Columbia.	1	do.[34]	8 ft. x 36 in.	Closed with classifier	Forged steel	5	75	12.2	18.2 31.1	+1 in −¼ in	1.3 39.4	+20 −200
	1	Tube[36]	5 x 8 ft	Closed with cones	Cast iron	2½	225	5.0			27.4 25.9	+50 −200
	1	Ball mill[34]	6 x 4½ ft	Closed with classifier	Forged steel	5	100	7.3	18.2 31.1	+1 in −¼ in	2.5 36.4	+20 −200
	1	do.[36]	4 x 8 ft	Closed with cones	Cast iron	2½	140	4.0			7.8 36.6	+50 −200
	1	do.[36]	6 ft. x 16 in.	Closed with classifier	Forged steel	1¾	133	3.8			6.0 48.9	+50 −200
Golden Belt, Ariz	1	do	4 x 5 ft	do	Cast iron	4		2.0		−1 to −1½ in	100	−48
Golden Turkey, Ariz	1	do	4 x 6 ft	do	Forged steel	3		3.1		−⅜ in	100	−40
Mammoth, Mont	1	do	6 x 4½ ft	Closed with classifier and unit cell.				2.0±				
Boss-Tweed, Mont	1	do	7 x 6 ft	Closed with classifier				3.3				
Mogollon Consolidated, N. Mex.	1	do	6 x 6 ft	do	Cast iron	3		6.5	100	−⅜	60–75	−150
Peach, Ariz	1	Rod mill										
Golden Anchor, Idaho	1	Ball mill	4½ x 6 ft	Closed with jig and classifier.	Forged steel	3		3.1		−½		−30
Big Jim, Ariz	1	do	6 x 4½ ft	do	Chrome steel			2.5	100	−1	80–82	−200
Kirkland Lake Gold, Ontario.	1	do	8 ft. x 30 in.	Open	Forged steel	5		6.73	31.6 1.6	+1, −2 in −150 mesh	2.0 28.3	−3, +10 −200, +300
	1	Tube mill	5½ ft. diam.	Closed with classifier and bowl.	do	1½	115	6.06	2.0 28.3	−3, +10 mesh. −200, +300	0.5 62.1	−20, +40 −200, +300
Coniaurum, Ontario	2	Tube mills in parallel.	5 x 16 ft	Closed with agitators and bowl classifier.	do	2		14.0	11.1 11.6	+4 mesh −200 mesh	[37] 10.0 [38] 60.0	[37] −65 [38] −200
Tom Reed, Ariz	1	Ball mill	6 x 6	Closed with primary classifiers.		5		12	2.5 2.2	−3, +2 in −200 mesh	0.84 3.99	+14 −200
	1	do	6 x 5									
	2	do	do	Closed with secondary classifiers.		2½		12	0.84 3.99	+14 −200	5.67 67.26	+65 −200
Black Hawk, N. Mex	2	do	5 ft. conical	Closed with classifiers		3		4.5±				
	1	do	6 x 6 ft	do		2		4.5±			50	−200
Siscoe, Quebec	2	do	6 x 4½	do	Chrome steel	5	322	16.2	1.8	−1 in. +¾ in.	12.2	+65
	1	do	5 x 5	do	Special metal	4	355	5.2	5.2	−32.5 mesh	48.9	−325
	1	do.[39]	4 x 4	do	Chrome steel	4						
Hollinger, Ontario, 1934.	7[34]	Rod mills	7 x 15 ft	Open	Carbon steel	3 diam. x 14 ft. long.		220	100 51.4	−.525 in −4 mesh	1.2 37.2	−6, +8 −200
	10[35]	Tube mills	6 x 16 ft	Closed with classifier	Flint pebbles	3–4–5	100	80	1.2 37.2	−6, +8 −200	0.8 64.9	+35 −200
	2[35]	do	5 x 20 ft	do								
	3[40]	do	do	Closed with classifier thickeners.	Forged steel	1⅜					.97 7.48 91.38	+150 +200 −200

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[34] Primary.

[35] Secondary.

[36] Regrind mills.

[37] Sands to agitators.

[38] Slimes to filtration.

[39] Regrind amalgamation tailing.

[40] Concentrate regrinding.

TABLE 57.—*Gold-milling data*—Continued

3. FINE-GRINDING PLANT—continued

Mine	Grinding and classification							New feed per hour, tons	Skeleton-size analysis of—			
	Mills			Circuit	Grinding media		Circulating load, per cent		Feed		Product	
	Number	Kind	Size		Kind of balls	Size of balls, inches			Per cent	Inch or mesh	Per cent	Mesh classifier overflow
Golden Cycle, Colo., (1) average Cripple Creek ore.[41]	3	Schmidt comminutors.	6 x 6 ft	Closed with 4½-mesh screens.		−½			6.2	+3/8 in	14.7	+8
									76.1	+8 mesh	10.4	−200
									0.5	−200 mesh		
	5	Chilean mills.[42]	6 ft	Closed with blankets, bowl, tray, and agitators.					14.7	+8 mesh	4.0	+8
									10.4	−200	11.8	−200
Golden Cycle, Colo., (2) low-grade Cripple Creek ores and complex sulfide custom ores.	2 in parallel.[36]	Ball mill	6 x 6 ft	Closed with classifier					12.6	+20 mesh	0.1	+80
									26.6	−200 mesh	84.0	−200
Lone Pine, Idaho	1	do	4½-ft. x 32 in.	Closed with screen and classifier.	Cast iron			1.8			100	−40
Orogrande-Frisco, Idaho.				Screens and rake classifiers separate sand and slime.							29.2	+8
											46.2	−8, +40
											7.8	−200

4. AMALGAMATION AND CONCENTRATION

Mine	Amalgamation					Concentration						
	Method of recovery	Feed			Recovery by amalgamation, percent	Method	Feed				Ratio of concentration	Recovery, percent
		Source	Skeleton-size analysis				Source	Skeleton-size analysis		Solids, percent		
			Percent	Mesh				Percent	Mesh			
Porcupine-United, Ontario.	Traps	Various				Blankets	Plates			32–36		
	Plates	Ball mill			[43] 33	Shaking table	Classifier overflow	65	−200			[43] 15.6.
	Barrel	Traps, blanket concentrate.			[43] 35							
Argonaut, Calif.	Plates	Stamps	100	−0.0277 (inch)	64.89	Spitzlutte vanners.	Plates, Spitzlutte	2.3	+35		41–1	24.14.
								4.22	+65			
	Traps	Plates	9.97 15.60	+35 −200		Callow cone	do	6.06	+200			
Alaska-Juneau, Alaska.	Barrel	Re-concentrating tables.			60.82	Hand-sorting, classifiers, v tanks, shaking tables.	Crushing	100	+4 in			
							Rolls and ball mills.	100	−7 mesh			17.69.
Atlanta, Idaho	Plate and traps	Ball mill and classifier.	3 33 100	+20 +35 −10	59.68 gold, 66.21 silver	Flotation	Classifier overflow	80	−200	27	55–1	29.87 gold, 28.77 silver.
Talache, Idaho	Traps, plates, pans.	Traps, table concentrate.			90	Shaking tables	Traps and plates					
Pilgrim, Ariz.	Barrels	Flotation concentrate.			84 gold	Flotation	Ball mill and classifier overflow.	53	−200	45	426–1	7.7.
Mossback, Ariz.	Launders, pan amalgamator.	Classifier			60	do	Launder					
		Flotation concentrates.										
Strawberry, Mont.	Plates and Den-	Stamps		24	[44] 40	do	Plate tailing	100	−24			[44] 41.

[36] Regrind mills. [41] Ore roasted before cyanidation and amalgamtion. [42] For grinding roasted ore. [43] Percent of total gold in mill heads. [44] On oxidized ore.

TABLE 57.—*Gold-milling data*—Continued

4. AMALGAMATION AND CONCENTRATION—continued

Mine	Amalgamation					Concentration						
	Method of recovery	Feed			Recovery by amalgamation, per cent	Method	Feed				Ratio of concentration	Recovery, per cent
		Source	Skeleton-size analysis				Source	Skeleton-size analysis		Solids, per cent		
			Per cent	Mesh				Percent	Mesh			
Cardinal, Calif.						Shaking tables	Ball-mill screen	100	−1/8 in	18–20	32–1	40.
						Flotation	Table middling via thickener.	4.5 60.7	+100 −200	40–45		50.
Spring Hill, Mont.						do	Classifier overflow	3.8 60.6	+80 −200	19	12–1	80–87.
Premier, British Columbia.						do	Overflow from cones.[34] do.[45]	1.3–2.5 36.4–32.4 2.5–6.0 36.4–48.9	+20 −200 +50 −200		8.44–1	93.35 gold, 86.29 silver.
Golden Belt, Ariz.						do	Classifier overflow	100	−48		14.4–1	96 gold, 80 silver.
Golden Turkey, Ariz.						do	do	100	−40		15.5–1	90 gold.
Mammoth, Mont.						Flotation shaking tables.					10–1	92.
Boss-Tweed, Mont.						Flotation					24–1	85.
Mogollon Consolidated, N. Mex.						do	Classifier overflow	60–75	−150		82.5–1	81 gold, 84 silver.
Peach, Ariz.						do					36–1	44.4.
Golden Anchor, Idaho.	Barrel	Gravity concentrates.			75.64 gold	Jig	Ball mill				272–1	17.39 gold.
						Classifier	Jig					
						Shaking table	Jig concentrate					
						Mat table	Flotation tail					
						Flotation	Jig tail Classifier overflow					
Yellow Aster mine, Calif.	Plates	Stamps		−30								

Yellow Aster tailing.						Rake classifiers	Stamp mill and old tailing.	Sands: 32.0 4.4 Slimes: 2.5 63.7	 +48 −200 +65 −200		
Siscoe, Que	Barrel	Cones Blankets			63.43 10.06	Hydraulic cones Corduroy blankets.	Ball mill Classifier overflow	12.2 48.9	+65 −325		
Hollinger, Ont						Shaking tables	Classifier Overflow from grinding section.	0.8 64.9	+35 −200	43.5	6–1
Golden Cycle, (1) Colo.	Amalgam pans	Blanket concentrate of roasted ore.			25.33	Blanket tables and flotation.	Chilean mill Classifier overflow	4.0 11.8	+5 −200		
Lone Pine, Idaho	Traps and barrels; mercury fed to ball mill.	Secondary crusher.				Hydraulic classifier shaking tables.	Ball-mill screen trap overflow.				
Orogrande-Frisco, Idaho.						Jigs Tables	Screen undersize Jig concentrates	100	−1/16 in.		

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[34] Primary.

[45] Secondary or regrind.

TABLE 57.—*Gold-milling data*—Continued

5. CYANIDATION

Mine	Skeleton-size analysis		Solids per-cent	Enters—	Pulp circuit [46]	Solutions					Recovery by cyanidation, percent
	Percent	Mesh				Strength, pounds NaCn per ton	Circuits [46]		Clarification	Precipitation	
							Barren	Pregnant			
Big Jim, Ariz.	80–82	−200	15	No. 1 or primary thickener.	Classifier overflow; No. 1 thickener; 3 agitators in series; No. 2 thickener of C. D.	2.2	Tank classifier overflow C. D.	No. 1 thickeners overflow clarifier, precipitation.	Leaf filter	Vacuum to precipitation press.	97–98
Kirkland Lake Gold, Ontario.	32.0 59.8	−150 −300	25	do	Classifier overflow; No. 1 thickener; No. 1 agitator bowl; bowl overflow; No. 2 agitator; No. 3 agitator; No. 2 thickener of C. D.	Grinding 0.9; other 0.7.	Tank; No. 5 thickener of C. D.	do	do	do	89.27
Coniaurum, Ontario.	11.1 11.6	+4 −200		Rake classifier.	Classifier overflow; 2 agitators in series with bowl classifier; tray thickener; filters.	1.25	Filters mill solution tank.	Tray thickener overflow clarifier precipitation.	Sand filter	do	96.35
Yellow Aster, Calif.; tailings.	Sands: 32.0 4.4	+48 −200		Eight 650-ton vats.	Vats; discharged to dam after leaching.	.032	Add to top of vats.	Filtrate to precipitation.	Leaf filter	do	
	Slimes: 2.5 63.7	+65 −200		3 thickeners	Thickener; 3 agitators, 3 filters.	.024					
Tom Reed, Ariz.	5.67 67.26	−65 +100 −200	63–72	No. 1 or primary thickener.	Classifier overflow; No. 1 thickener; 4 agitators; C. D.	2.20	Add to No. 5 thickener and to ball mill classifier.	No. 1 thickener overflow; clarifier; precipitation.	do	Precipitation press.	94.5 (gold) 63.8 (silver)
Black Hawk, N. Mex.	50.0	−200		do	Classifier overflow; No. 1 thickener; Pachucas; C. D.; filter.		Add to ball mill and Pachuca tanks.	No. 1 thickener overflow; clarifier; zinc boxes.	do	Zinc boxes	88–93
Siscoe, Quebec	15.0 54.5	+65 −200		Agitator	Blanket tail; No. 1 thickener; 3 agitators in series; filter; concentrates reground and returned to ball-mill circuit.		Add to ball mills, filters and repulper, etc.	No. 1 thickener overflow; vacuum precipitation.	do	Vacuum precipitation press.	24.02

Hollinger, Ontario, table tailing.	----------	----------	------	No. 1 or primary thickener.	Tables; 12 primary tray thickeners - agitators [47]; part of underflow to C. D.; part to primary filters-secondary filters.	0.75	Tank to rod mills, tube mills, tables, thickener; agitator C. D.	No. 1 thickener overflow; clarifier; precipitation.	Clarifying presses.	do	95.8
Hollinger, Ontario, table concentrate.	91.38	[48]—200	------	Special agitator.	Special agitators; then joins tail underflow from primary thickener.	1.4	Primary filter to mill tank.[49]	do	do	do	95.8
Golden Cycle, Colo., (1).	4.0–5.0 84.0	+100 −200	7.0	(50)	Bowl slimes to thickener to agitator to thickener to agitator to leaf filters. Bowl sands to leaching vats.[50]	2.0–0.7	First tray thickener, Chilean mills.	Vats and secondary thickener to vacuum to precipitation.	Sand filter	do	71.48
Orogrande - Frisco, Idaho.	Sands Slimes	+1/16 inch −1/16 inch	-----	Leaching vats.	Classifier sands to strong solution leach, to aeration, to second strong leach, to weak solution leach, to water wash.	Strong solution, 2.0	Screen to leach vats, 62 hours.	Leach vats to clarification, to precipitation.	do	do	----------

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[46] C. D. countercurrent decantation section.

[47] 4 parallel groups of 5 each in series.

[48] After regrinding.

[49] Solution used throughout the mill; water used only for final filter wash.

[50] Feed comprises blanket tailing to bowl classifier, to thickener, and flotation concentrate from low-grade Cripple Creek and complex sulfide custom ore.

TABLE 57.—*Gold-milling data*—Continued

6. PRODUCTION AND COST DATA. A.

Mine and year	Average ore treated per day, tons	Average value of heads, per ton	Over-all recovery, percent	Costs per ton of ore milled							
				Crushing	Grinding, screening, classifying	Amalgamating	Concentrating	Cyaniding	Disposal of tailing	Miscellaneous	Total
Porcupine-United, Ontario, 1930–31	25	[2] $11.00	83.6	$0.384	[51]$0.771	[52]$0.626		None		$0.070	$1.851
Argonaut, Calif., 1929	246	[2] 6.07	89.03	See following section of table, 6–B							.960
Alaska-Juneau, Alaska, 1929	[5] 10,718 [6] 5,078	[5] 1.100 [6] 1.919	78.51	.052	.163	.015	$0.035	None	$0.051		.316
Atlanta, Idaho, April 1934	196	[8] 16.34	89.55 gold, 94.58 silver	.102	.490	.082	.262		$0.057	.169	1.162
Talache, Idaho, Sept. 1936	119	[8] 7.00	90	.130	.345	.062	.048			.055	.640
Pilgrim, Ariz., 1936	90	[8] 9.25 gold only	91.7 gold, 73.0 silver	See following section of table, 6–B							1.049
Cardinal, Calif., June 1937	186	[8] 9.45	90	do							1.240
Spring Hill, Mont., July 1929 to March 1930.	153	[2] 6.46	82	.162	.290		.258		.052	.245	1.007
Premier, British Columbia, 1931	490	[2] 6.30 plus 5.93 oz. silver	93.35 gold, 86.29 silver	.162	.465		.231			.209	1.067
Golden Belt, Ariz., Jan. 1936	[6] 48	[6] [8] 7.35 gold only	96.0 gold, 80.0 silver	See following section of table, 6–B							1.020
Golden Turkey, Ariz., year ended Oct. 31, 1936.	[6] 70	[6] [8] 5.59	90+	do							1.090
Mammoth, Mont., 1935	50	[8] 9.10	92	do							1.680
Mogollon Consolidated, N. Mex., May 1936.	150	[8] [25] 8.25	81 gold, 84 silver	.370	.360		.440		.200	.180	1.550
Peach, Ariz., 1936	250–300	[8] 3.15	44.4	See following section of table, 6–B							.359
Golden Anchor, Idaho, 1937	50	[8] 24.88 gold, 2.58 oz. silver	93.03 gold, 80.71 silver	do						[53] 890	[53] 2.700
Big Jim, Ariz., July 1933	53		97–98	.104	.686			$0.802		.233	[54] 1.825
Kirkland Lake Gold, Ontario, 1930	143	11.38	89.27	.145	.642			.532		.070	1.389
Coniaurum, Ontario, 1930	335	6.00	96.35	.346	.202			.314	.015	.062	.939
Yellow Aster, Calif., mine and tailings, 1 month 1936.	190 ore, 910 tails.			$0.233					.029		[55] .262
Tom Reed, Ariz., 1934–36	290	[8] 12.19 gold, 0.52 silver	94.5 gold, 63.85 silver	See following section of table, 6–B							1.656
United Eastern, Ariz., 1922–25	260 (1918) 325 (1922)	19.87 [1]	96.59	.055	.825			.725	.025	.364	1.994
Black Hawk, N. Mex., 4 months 1936	100	9 to $12 [8] gold and silver [4]	88–93	.357	.325			.822	.239	.147	1.890
Siscoe, Quebec, Dec. 1936 to Feb. 1937	[56] 561, [57] 501	[4] 17.52	97.51	[58] .211	[59] .295	.035	.037	.268	.008	.131	.985
Hollinger, Ontario, 1934	5,236	[28] 7.68		.081	.146		.061	.177	.017	.114	.596
Golden Cycle, Colo., 1929	818	[1] 10.38	96.81	.202	.270	.022		.470	.068	.546 [60] .635	2.214
Lone Pine, Idaho, Sept. 1936	43.5			See following section of table, 6-B							.918
Orogrande-Frisco, Idaho, 1937	500–600	1.50 (approx.)	80–83	do							[61] .318

6. PRODUCTION AND COST DATA. B

Mine and year	Costs per ton of ore milled								
	Labor and supervision	Power and water	Supplies					Miscellaneous	Total
			Crushing	Grinding and classifying	Amalgamating	Concentration	Cyaniding		
Porcupine-United, Ontario, 1930–31	$1.072	$0.405	$0.035	$0.269			None	$0.070	$1.851
Argonaut, Calif., 1929	.230	.190				$0.380	None	.160	.960
Alaska-Juneau, Alaska, 1929	[62].108	[41].029		[62] [64].099			None		[62].236
	[63].144	.039		[63] [64].133			None		.316
Atlanta, Idaho, Apr. 1934	.476	.264		[64].422			None		1.162
Talache, Idaho, Sept. 1936	.225	.135		[64].205			None	.075	.640
Pilgrim, Ariz., Oct.–Dec. 1936	.370	.324		[64].355			None		1.049
Cardinal, Calif., June 1937	.370	.280		[64].590	None		None		1.240
Spring Hill, Mont., 1929–30	.426	.208	.050	.072		.118		.133	1.007
Premier, British Columbia, 1931	.349	.274	.040	.262		.082		.060	1.067
Golden Belt, Ariz., Jan. 1936	.570	.310		[64].140					1.020
Golden Turkey, Ariz., year ended Oct. 31, 1936	[63].620	.270		[64].270				.030	[65] 1.090
Mammoth, Mont., 1935	.560	.400		[64].350				.370	1.680
Mogollon Consolidated, N. Mex., May 1936	.800	.265	.066	.137		.214		.068	1.550
Peach, Ariz., 1936	.249	.042				.068			.359
Golden Anchor, Idaho, 1937	.570	.720		[64].520				.890	2.700
Big Jim, Ariz., July 1933	.779	.668		.135			$0.215	.028	1.825
Kirkland Lake Gold, Ontario, 1930	.429	.368	.040	.234			.256	.062	1.387
Yellow Aster, Calif., mine and tailings	.076	.145		[64].084				.024	.329
Tom Reed, Ariz., 1934–36	.390	.526		[64].565				[66].175	1.656
Black Hawk, N. Mex., 4 months, 1936	.708			[64] 1.120				.062	1.890
Golden Cycle, Colo., 1929	.694	.255	.070	.113	$0.006	[67].383	.355	.338	2.214
Lone Pine, Idaho, Sept. 1936	.608	.065		.179				.066	.918
Orogrande-Frisco, Idaho, 1937	.106	([68])				[64] 0.173		[69].039	[70].318

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[1] Vary, R. A., Amalgamation Practice at the Porcupine United Gold Mines (Ltd.), Timmins, Ontario: Bureau of Mines Inf. Circ. 6433, 1931, 5 pp.

[2] Gold $20.67 per ounce.

[4] Bradley, P. R., Milling Practice at the Alaska-Juneau Concentrator: Bureau of Mines Inf. Circ. 6236, 1930, 16 pp.

[5] Crude ore.

[6] Sorted ore.

[8] Gold $35.00 per ounce.

[25] Gold 40 percent, silver 60 percent of total value.

[28] Gold at $32.16 per ounce.

[51] Grinding only.

[52] Classifying, screening, conveying, and refining.

[53] Includes $0.13 express and mint charges, $0.15 freight and smelter charges on concentrates, and $0.61 overhead.

[54] Rental of mill $0.612 additional.

[55] Costs include amalgamation and cyanidation (mine plus tailing).

[56] Tons crushed and sorted.

[57] Tons cyanided.

[58] Includes sorting.

[59] Includes regrind circuit $0.020.

[60] Roasting.

[61] Except power and heat (power, light, and heat for mining and milling, $0.0578 per ton).

[62] Per ton trammed.

[63] Per ton sorted ore milled.

[64] All supplies.

[65] Not including $0.15 depreciation.

[66] Repairs.

[67] Roasting supplies.

[68] Power, light, and heat for mining and milling, $0.0578.

[69] Industrial insurance, miscellaneous expense, and depreciation.

[70] Except power and heating.

COPPER ORES

Chapman [88] has discussed the concentration of copper ores in earlier Bureau of Mines publications. Bulletin 392 is a comprehensive work containing operating and cost data in permanent printed form and is available for consultation in most large technical libraries or may be purchased from the Superintendent of Documents, Government Printing Office, Washington, D. C., for 15 cents as long as the supply lasts. A résumé of the earlier discussion and data touching only those points of special significance to the purpose of the present bulletin is given in the pages immediately following.

The four major steps in the production of marketable copper are mining, concentrating, smelting, and refining. In a few instances, however, leaching takes the place of concentrating, smelting, and refining, * * *. At present, although considerable leaching and direct-smelting ores are produced, the bulk of the copper ore mined is concentrated. * * *.

From 1912 to 1932 the milling of copper ores as practiced in the larger concentrators has changed to such an extent that comparatively few of the machines in use at the beginning of the period remain in service today. Primary and secondary crushing by machines of the Blake and gyratory types and intermediate and fine crushing by rolls has survived, but in the grinding field the development of pebble-mill grinding, the substitution of balls for pebbles, and the parallel development of drag-type classifiers have all but eliminated Chilean and Huntington mills * * *. In the concentrating field, * * * machines which effected separations on the basis of difference in specific gravity between copper and gangue minerals have been almost completely replaced by flotation equipment. In the Lake Superior district jigs and tables have, of course, been retained, and in a few concentrators which treat sulphide copper ores tables have been retained owing to unusual conditions at the plants or the smelters that treat the mill concentrates.

The flotation process, which was responsible for the almost complete change in equipment * * *, has also undergone marked changes since its introduction in large-capacity concentrators * * *. Flotation, when first introduced between 1913 and 1916, was used primarily to reduce losses of copper in the fine tailings of gravity plants. From an accessory to gravity methods, flotation very rapidly became a major process and finally, from 1923 to 1927, all but eliminated the gravity method in the treatment of low-grade sulfide copper ores.

The rapid development of ball-mill grinding * * * must also be attributed to the adoption of the flotation process * * *, since it was the incentive for developing grinding methods which produced considerable copper minerals too finely divided for successful recovery by existing gravity methods * * *.

The benefits derived from the adoption of flotation methods * * * include: (1) Increased recoveries of copper; (2) simplification of the milling process with resultant decrease in the number of stages of treatment and the number of machines required; (3) production of higher-grade concentrates by the elimination of insoluble and pyrite which had formerly been included in the concentrates produced by gravity methods; (4) production of concentrates other than copper concentrates from plants essentially copper plants—for example, the production of pyrite concentrates by the Britannia Mining Co., the production of zinc concentrates at the Flin Flon and Sherritt Gordon plants, and the production of nickel-copper concentrates at the concentrator of the International Nickel Co., all by [differential] flotation methods; (5) decreased first cost of concentrating plants per ton of daily capacity and decreased operating and repair costs per ton of ore treated; (6) decreased costs of milling and mining operations and improved metallurgical results, permitting lower-grade copper ores to be mined and treated and thereby increasing the available ore reserves of many properties; (7) general adoption of flotation process, influencing the development of the reverberatory furnace, which generally has replaced the blast furnace in smelting operations, with resultant benefits in the cost of copper smelting.

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[88] Chapman, Thomas G., Concentration of Copper Ores in North America: Bureau of Mines Bull. 392, 1936, 169 pp. Trends in Practice and Costs of Concentrating Copper Ores: Bureau of Mines Inf. Circ. 6792, 1934, 45 pp.

TYPES OF ORES AND METHODS OF TREATMENT

From the milling standpoint, copper ores may be classified roughly as follows:

1. Ores in which the copper is present in the native or metallic form, represented principally by the ores of the Lake Superior district. These ores respond to gravity concentration of the coarser particles of copper, and the finer particles may be recovered by flotation or, as at Calumet & Hecla Conglomerate mill, the sand portion of the gravity-plant tails may be leached and the slime portion may be treated by flotation.
2. Ores containing coarse sulfides of copper and oxidized ores containing cuprite, which respond to both gravity and flotation methods; these may be concentrated by gravity methods to recover the coarser particles and the tailing from the gravity machines by flotation; or fine-grinding and flotation only may be used. In most modern plants the gravity step is omitted.
3. Ores in which the copper occurs in the form of fine sulfide mineral particles disseminated throughout the gangue and which are treated by fine grinding and flotation. These ores are represented notably by the so-called porphyry-copper ores.
4. Oxidized ores in which malachite, azurite, and chrysocolla are the predominant copper minerals that do not respond either to gravity or simple flotation methods. Sulfide-filming of the carbonates followed by flotation of the filmed minerals, or direct leaching followed by precipitation of the copper from the solution may be employed for such ores.
5. Ores containing sulfide copper minerals together with enough sulfides of zinc, lead, iron, or other metals to warrant use of differential flotation to separate them and to make a separate concentrate of each.

As with gold and other ores, details of practice vary because of differences in the ores or on account of economic considerations. Five figures are presented to illustrate in a general way the concentrating methods employed for treating the different types of ores.

Figure 150[89] is the flow sheet of one unit of the gravity concentration section of the Calumet & Hecla Conglomerate mill. The sand tailing from the mill is treated by ammonia leaching and the slime tailing by flotation. Since metallic copper is malleable, it cannot be broken and pulverized as can the more friable minerals, and after first picking out the larger lumps or nuggets of copper by hand, crushing is done by steam stamps; pebble mills instead of ball mills are employed for grinding because of the abrasive qualities of the gangue.

Figure 151[90] is the flow sheet of one section of the Cananea Consolidated Copper Co. mill as it was in 1929. This is a simple straight-flotation process that replaced an earlier combined gravity and bulk-

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[89] Benedict, C. Harry, Milling Method and Cost at the Conglomerate Mill of the Calumet & Hecla Consolidated Copper Co.: Bureau of Mines Inf. Circ. 6364, 1930, 22 pp.

[90] Tye, A. T., Milling Methods and Costs at the Concentrator of the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico: Bureau of Mines Inf. Circ. 6261, 1930, 21 pp.

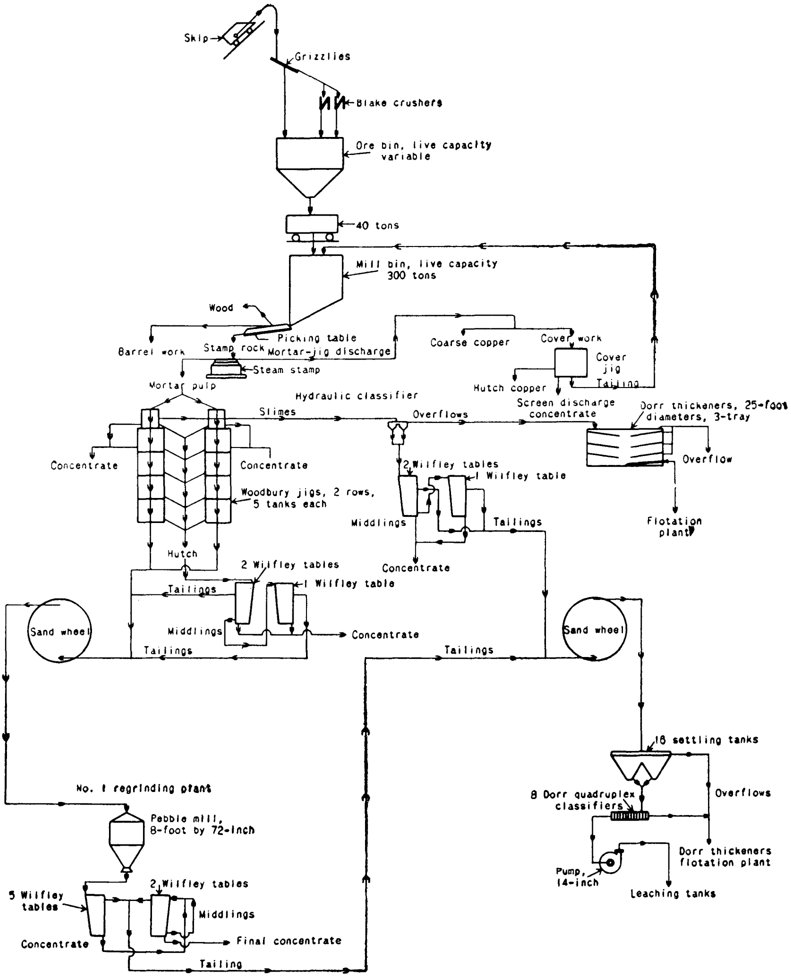
Skip
Grizzlies
Blake crushers
Ore bin, live capacity variable
40 tons
Mill bin, live capacity 300 tons
Wood
Picking table
Barrel work
Stamp rock
Mortar-jig discharge
Coarse copper
Cover work
Cover jig
Hutch copper
Tailing
Steam stamp
Mortar pulp
Hydraulic classifier
Screen discharge concentrate
Slimes
Overflows
Dorr thickeners, 25-foot diameters, 3-tray
Overflow
Concentrate
Concentrate
Woodbury jigs, 2 rows, 5 tanks each
2 Wilfley tables
1 Wilfley table
Middlings
Tailings
Flotation plant
Concentrate
Hutch
2 Wilfley tables
1 Wilfley table
Tailings
Middlings
Concentrate
Sand wheel
Tailings
Tailings
Sand wheel
18 settling tanks
No. 1 regrinding plant
Pebble mill, 8-foot by 72-inch
8 Dorr quadruplex classifiers
Overflows
Dorr thickeners flotation plant
Pump, 14-inch
Leaching tanks
5 Wilfley tables
2 Wilfley tables
Middlings
Final concentrate
Concentrate
Tailing

FIGURE 150.—Gravity flow sheet of Calumet & Hecla (Mich.) Conglomerate mill.

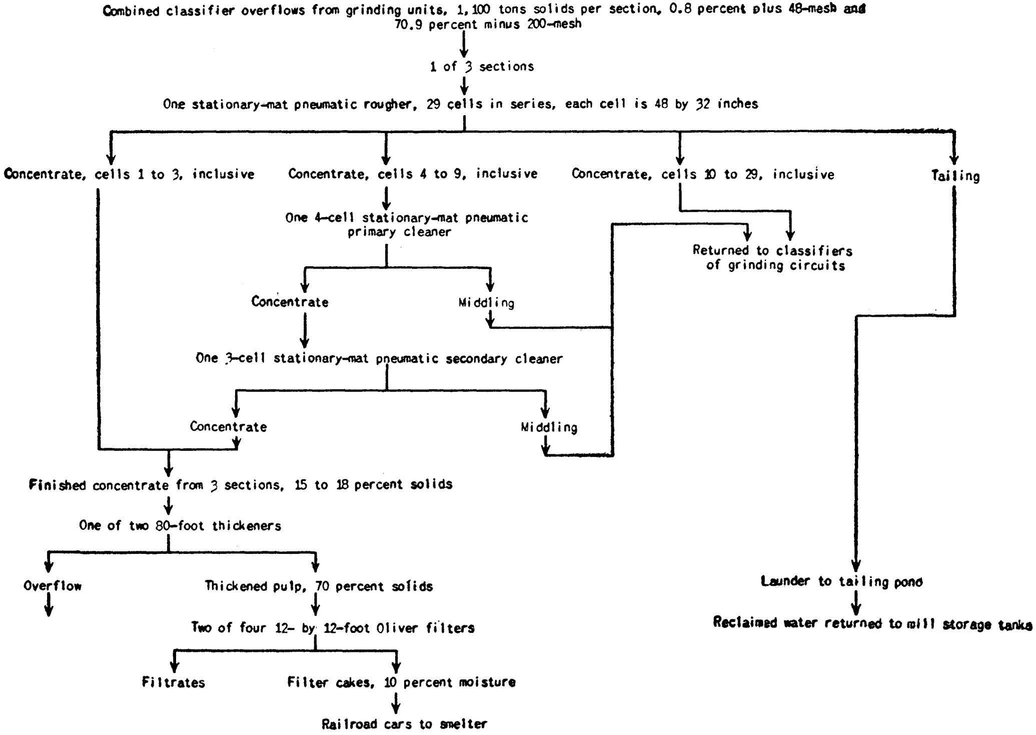
Combined classifier overflows from grinding units, 1,100 tons solids per section, 0.8 percent plus 48-mesh and 70.9 percent minus 200-mesh
1 of 3 sections
One stationary-mat pneumatic rougher, 29 cells in series, each cell is 48 by 32 inches
Concentrate, cells 1 to 3, inclusive
Concentrate, cells 4 to 9, inclusive
Concentrate, cells 10 to 29, inclusive
Tailing
One 4-cell stationary-mat pneumatic primary cleaner
Returned to classifiers of grinding circuits
Concentrate
Middling
One 3-cell stationary-mat pneumatic secondary cleaner
Concentrate
Middling
Finished concentrate from 3 sections, 15 to 18 percent solids
One of two 80-foot thickeners
Overflow
Thickened pulp, 70 percent solids
Two of four 12- by 12-foot Oliver filters
Filtrates
Filter cakes, 10 percent moisture
Railroad cars to smelter
Launder to tailing pond
Reclaimed water returned to mill storage tanks

FIGURE 151.—Flow sheet of No. 1 section, Cananea (Mexico) concentrator.

flotation method. Table 58 summarizes the differences between results obtained from the earlier and later methods.

TABLE 58.—*Comparison of combined gravity and bulk flotation with differential flotation at Cananea*

Flow sheet	Combined gravity and bulk flotation	Differential flotation
Period	Last half 1923	First half 1929
Cost of concentrating:		
Operation per ton	$0.62	$0.51
Repairs do	$.16	$.10
Total do	$.83	$.69
Ore concentrated dry tons	252,120	406,010
Ratio of concentration	2.57	6.81
Recovery:		
Copper percent	87.40	95.74
Silver do	88.21	96.19
Gold do	95.80	88.28
Power kw.-hr. per ton	15.8	15.2
Pine oil used per ton of ore treated pound	.61	.14
Xanthate used per ton of ore treated pound	------------	.086
Lime used per ton of ore treated pounds	------------	8.80
Cost of flotation reagents, including lime per ton	$0.0954	$0.0550
Copper in ore percent	1.95	3.29
Iron in ore percent	16.70	10.31
Copper in concentrates percent	4.38	21.44
Copper in tailings do	.29	.19
Iron in tailings do	5.20	6.60
Insoluble in concentrates percent	15.20	12.30
Moisture in concentrates percent	11.40	9.80
Cost of rods and balls per ton of ore treated	$0.0790	$0.0690
Copper produced in concentrates tons	4,286	12,777
Plus 48-mesh material in tailings percent	4.1	3.0
Plus 65-mesh material in tailings, cumulative percent	15.6	14.1
Minus 200-mesh material in tailings percent	61.1	60.2

Figure 152 is the flow sheet of the Miami Copper Co. concentrator as it was in 1932; *A* shows the crushing plant and *B* the grinding and flotation units. Figure 153 [91] is the flow sheet of the Miami concentrate re-treatment and filter plants. The ores are composed of chalcocite and pyrite with subordinate amounts of oxidized copper minerals disseminated mainly in a quartz-sericite schist.

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[91] Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Bureau of Mines Inf. Circ. 6573, 1932, 25 pp.

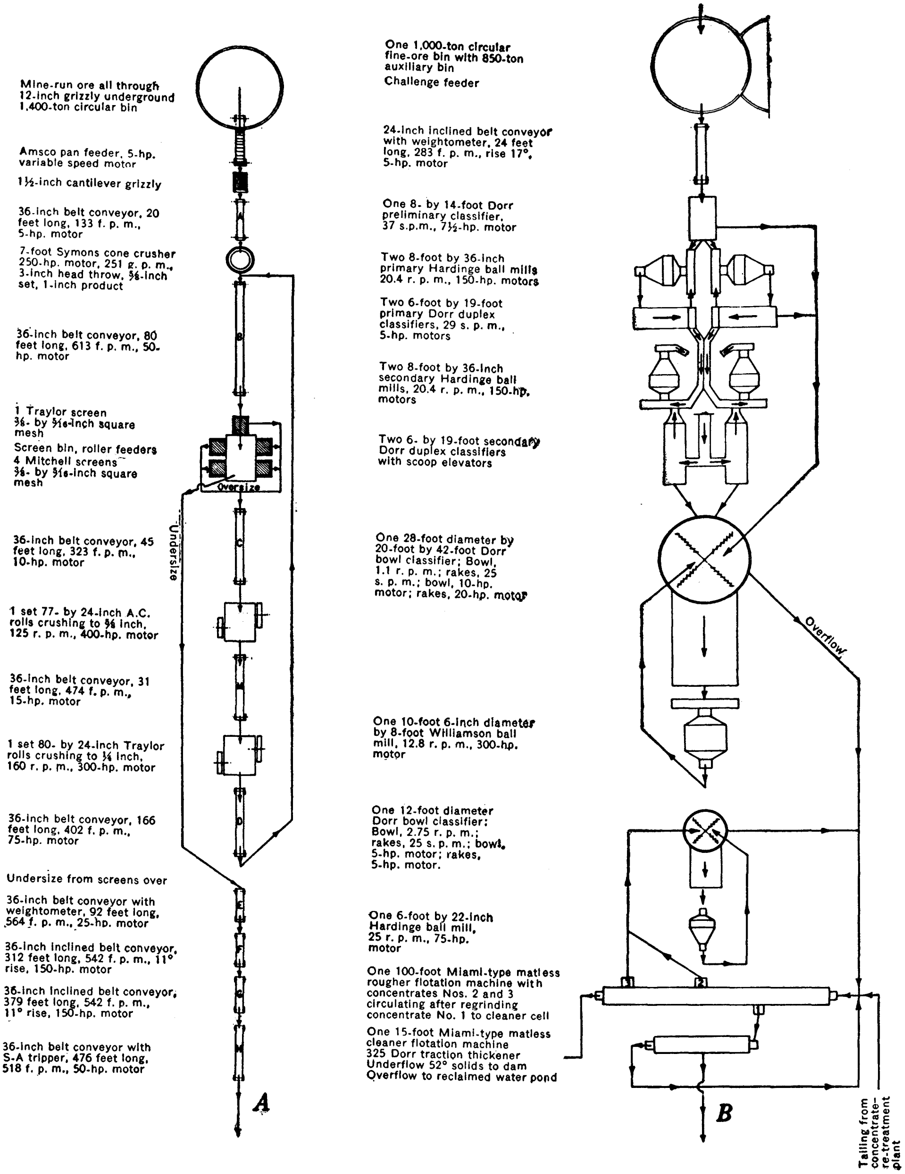
Mine-run ore all through 12-inch grizzly underground 1,400-ton circular bin
Amsco pan feeder, 5-hp. variable speed motor
1½-inch cantilever grizzly
36-inch belt conveyor, 20 feet long, 133 f. p. m., 5-hp. motor
7-foot Symons cone crusher 250-hp. motor, 251 g. p. m., 3-inch head throw, ⅝-inch set, 1-inch product
36-inch belt conveyor, 80 feet long, 613 f. p. m., 50-hp. motor
1 Traylor screen ⅜- by 5/16-inch square mesh
Screen bin, roller feeders
4 Mitchell screens ⅜- by 5/16-inch square mesh
Oversize
Undersize
36-inch belt conveyor, 45 feet long, 323 f. p. m., 10-hp. motor
1 set 77- by 24-inch A.C. rolls crushing to ⅝ inch, 125 r. p. m., 400-hp. motor
36-inch belt conveyor, 31 feet long, 474 f. p. m., 15-hp. motor
1 set 80- by 24-inch Traylor rolls crushing to ¼ inch, 160 r. p. m., 300-hp. motor
36-inch belt conveyor, 166 feet long, 402 f. p. m., 75-hp. motor
Undersize from screens over
36-inch belt conveyor with weightometer, 92 feet long, 564 f. p. m., 25-hp. motor
36-inch inclined belt conveyor, 312 feet long, 542 f. p. m., 11° rise, 150-hp. motor
36-inch inclined belt conveyor, 379 feet long, 542 f. p. m., 11° rise, 150-hp. motor
36-inch belt conveyor with S-A tripper, 476 feet long, 518 f. p. m., 50-hp. motor
A
B
C
D
E
F
G
H
One 1,000-ton circular fine-ore bin with 850-ton auxiliary bin
Challenge feeder
24-inch inclined belt conveyor with weightometer, 24 feet long, 283 f. p. m., rise 17°, 5-hp. motor
One 8- by 14-foot Dorr preliminary classifier, 37 s.p.m., 7½-hp. motor
Two 8-foot by 36-inch primary Hardinge ball mills 20.4 r. p. m., 150-hp. motors
Two 6-foot by 19-foot primary Dorr duplex classifiers, 29 s. p. m., 5-hp. motors
Two 8-foot by 36-inch secondary Hardinge ball mills, 20.4 r. p. m., 150-hp. motors
Two 6- by 19-foot secondary Dorr duplex classifiers with scoop elevators
One 28-foot diameter by 20-foot by 42-foot Dorr bowl classifier; Bowl, 1.1 r. p. m.; rakes, 25 s. p. m.; bowl, 10-hp. motor; rakes, 20-hp. motor
Overflow
One 10-foot 6-inch diameter by 8-foot Williamson ball mill, 12.8 r. p. m., 300-hp. motor
One 12-foot diameter Dorr bowl classifier: Bowl, 2.75 r. p. m.; rakes, 25 s. p. m.; bowl, 5-hp. motor; rakes, 5-hp. motor.
One 6-foot by 22-inch Hardinge ball mill, 25 r. p. m., 75-hp. motor
One 100-foot Miami-type matless rougher flotation machine with concentrates Nos. 2 and 3 circulating after regrinding concentrate No. 1 to cleaner cell
1
2
3
One 15-foot Miami-type matless cleaner flotation machine
325 Dorr traction thickener
Underflow 52° solids to dam
Overflow to reclaimed water pond
Tailing from concentrate-re-treatment plant
A
B

FIGURE 152.—Flow sheet of Miami Copper Co. (Arizona) concentrator (1932).

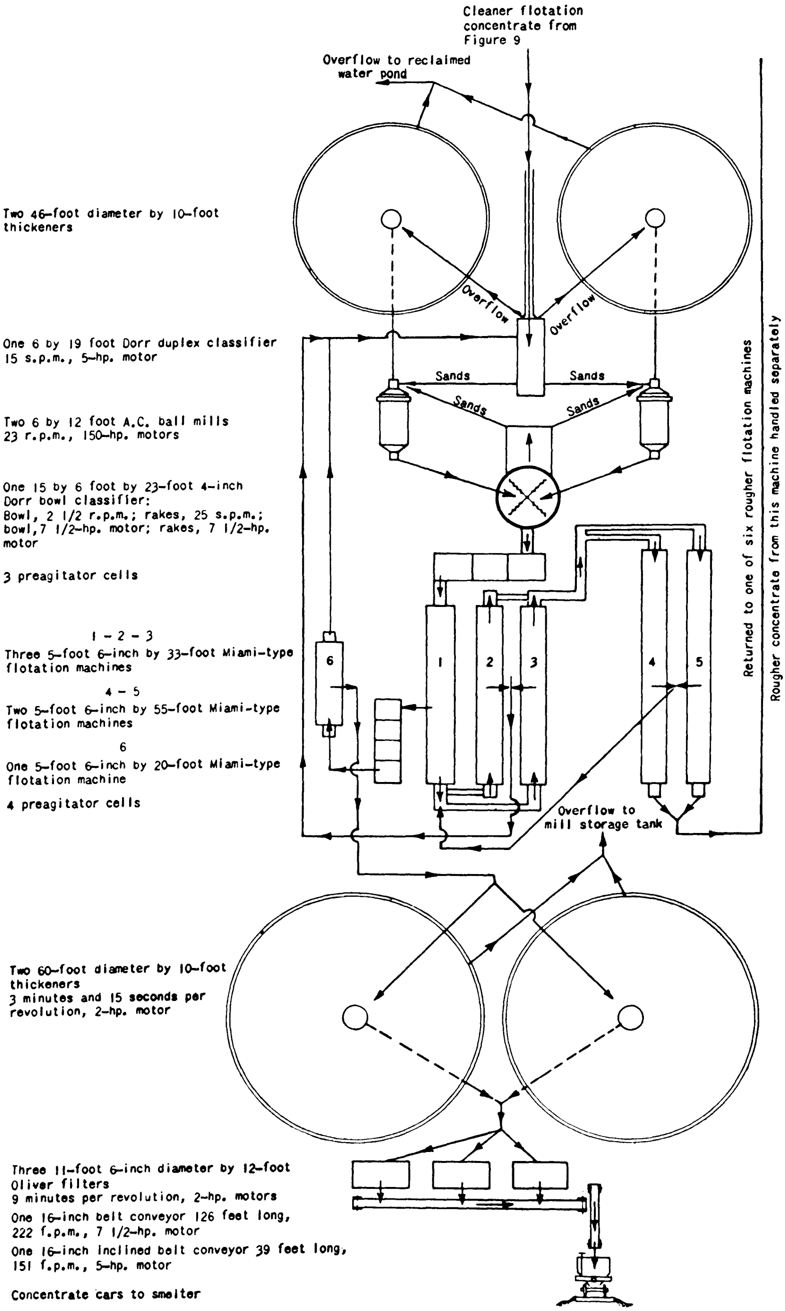
Cleaner flotation concentrate from Figure 9
Overflow to reclaimed water pond
Two 46-foot diameter by 10-foot thickeners
Overflow
Overflow
One 6 by 19 foot Dorr duplex classifier 15 s.p.m., 5-hp. motor
Sands
Sands
Sands
Sands
Two 6 by 12 foot A.C. ball mills 23 r.p.m., 150-hp. motors
One 15 by 6 foot by 23-foot 4-inch Dorr bowl classifier: Bowl, 2 1/2 r.p.m.; rakes, 25 s.p.m.; bowl, 7 1/2-hp. motor; rakes, 7 1/2-hp. motor
3 preagitator cells
1 - 2 - 3
Three 5-foot 6-inch by 33-foot Miami-type flotation machines
4 - 5
Two 5-foot 6-inch by 55-foot Miami-type flotation machines
6
One 5-foot 6-inch by 20-foot Miami-type flotation machine
4 preagitator cells
6
1
2
3
4
5
Overflow to mill storage tank
Returned to one of six rougher flotation machines
Rougher concentrate from this machine handled separately
Two 60-foot diameter by 10-foot thickeners 3 minutes and 15 seconds per revolution, 2-hp. motor
Three 11-foot 6-inch diameter by 12-foot Oliver filters 9 minutes per revolution, 2-hp. motors
One 16-inch belt conveyor 126 feet long, 222 f.p.m., 7 1/2-hp. motor
One 16-inch inclined belt conveyor 39 feet long, 151 f.p.m., 5-hp. motor
Concentrate cars to smelter

FIGURE 153.—Flow sheet of Miami (Ariz.) concentrate re-treatment and filter plants.

Figure 154 is a flow sheet of the flotation section of the Britannia concentrator.[92] Here "the flotation treatment comprises first the production of bulk concentrates which contain the chalcopyrite and pyrite with minor amounts of gold and silver. The bulk concentrates are dewatered and, after additional grinding, are again subjected to flotation. The latter operation produces finished copper concentrate, finished pyrite concentrate, and middlings which are re-treated."

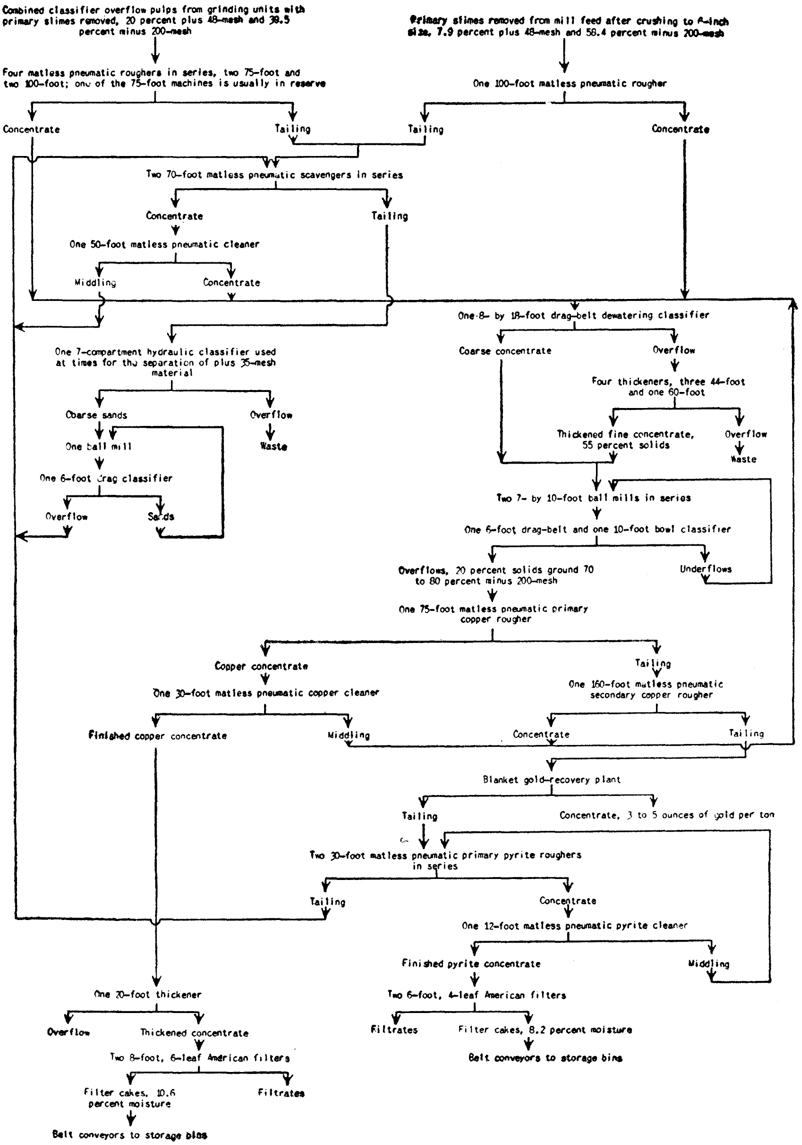
Combined classifier overflow pulps from grinding units with primary slimes removed, 20 percent plus 48-mesh and 39.5 percent minus 200-mesh
Primary slimes removed from mill feed after crushing to 8-inch size, 7.9 percent plus 48-mesh and 58.4 percent minus 200-mesh
Four matless pneumatic roughers in series, two 75-foot and two 100-foot; one of the 75-foot machines is usually in reserve
One 100-foot matless pneumatic rougher
Concentrate
Tailing
Tailing
Concentrate
Two 70-foot matless pneumatic scavengers in series
Concentrate
Tailing
One 50-foot matless pneumatic cleaner
Middling
Concentrate
One 8- by 18-foot drag-belt dewatering classifier
Coarse concentrate
Overflow
Four thickeners, three 44-foot and one 60-foot
Thickened fine concentrate, 55 percent solids
Overflow
Waste
One 7-compartment hydraulic classifier used at times for the separation of plus 35-mesh material
Coarse sands
Overflow
Waste
One ball mill
One 6-foot drag classifier
Overflow
Sands
Two 7- by 10-foot ball mills in series
One 6-foot drag-belt and one 10-foot bowl classifier
Overflows, 20 percent solids ground 70 to 80 percent minus 200-mesh
Underflows
One 75-foot matless pneumatic primary copper rougher
Copper concentrate
Tailing
One 30-foot matless pneumatic copper cleaner
One 160-foot matless pneumatic secondary copper rougher
Finished copper concentrate
Middling
Concentrate
Tailing
Blanket gold-recovery plant
Tailing
Concentrate, 3 to 5 ounces of gold per ton
Two 30-foot matless pneumatic primary pyrite roughers in series
Tailing
Concentrate
One 12-foot matless pneumatic pyrite cleaner
Finished pyrite concentrate
Middling
One 20-foot thickener
Two 6-foot, 4-leaf American filters
Overflow
Thickened concentrate
Filtrates
Filter cakes, 8.2 percent moisture
Two 8-foot, 6-leaf American filters
Belt conveyors to storage bins
Filter cakes, 10.6 percent moisture
Filtrates
Belt conveyors to storage bins

FIGURE 154.—Flow sheet of flotation at Britannia (B. C.) concentrator.

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[92] Munro, A. C., and Pearse, H. A., Milling Methods and Costs at the Concentrator of the Britannia Mining & Smelting Co. (Ltd.), Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6619, 1932, 24 pp.

DATA ON COPPER CONCENTRATORS

Table 59 summarizes cost and other data on a number of copper-ore concentrators.

The chief purpose of the table is to present typical milling costs (sec. 8). Sections 1 to 7, inclusive, give supporting data from which the reader may deduce the principal reasons for the variations in costs. Thus, the lowest costs are obtained at the large-capacity plants, principally due to the spread of fixed labor and other fixed operating charges over a greater tonnage. At most plants the cost of grinding is the largest single item except where relatively coarse grinding suffices to liberate the ore minerals. At the plants listed the grinding costs range from a minimum of 15 to a maximum of 52 percent of the total operating cost, the arithmetical average being 33 percent and the ordinary range from 25 to 44 percent. Also, it may be noted that, on the average, labor cost is about 30 percent and power and water supply cost about 26 percent of the total cost.

TABLE 59.—*Data on copper-ore concentrators*—Continued

1. GENERAL DATA—continued

Mine, reference, and year	General nature of ore treated; copper and associated minerals and rocks	Mill heads, copper, percent	General method of treatment
Matahambre, Cuba, 1930 [13]	Chalopyrite with pyrite in quartzite and shale	4. 55	Flotation (4 roughing stages. First-stage concentrate double-cleaned; all other concentrates and cleaner middlings returned to roughers).
Engels, Calif., 1929 [14]	Chalcopyrite and bornite in gangue of fine-grained tourmaline in quartz monzonite (Superior mine) and in altered diorite (Engels mine).	1. 524	Flotation (finished concentrates in first cell of 12-cell mechanical machine and tailing from last cell go to 6-cell pneumatic scavenger. Froth from cells 2 to 4 of mechanical machine cleaned once and from cells 5 to 7 cleaned twice. All other rougher froths returned to head of circuit).
Miami, Ariz., 1930 [15]	Chalcocite, pyrite, and subordinate oxidized copper minerals disseminated in quartz-sericite schist.	[2] 0. 716 [6] 0. 112	Bulk flotation of sulfides producing finished tailing; concentrate reground and treated by differential flotation depressing pyrite and producing final concentrate.
Britannia, British Columbia, July 1931.[16]	Chalcopyrite and minor chalcocite and bornite with considerable pyrite in chlorite schist.	1. 05	Bulk flotation of chalcopyrite and pyrite in 2 roughing stages. Separate primary rougher for slimes separated after primary crushing. Final tailing from secondary roughers. Primary rougher concentrate and cleaned secondary reground and treated by selective flotation.
Utah Copper, Utah, 1929–1930, Arthur and Magna plants.[17]	Copper sulfides and pyrite finely disseminated in monzonite porphyry, in quartzite, and in epidotized limestone.	0. 9945 (1929) 0. 9603 (Magna 1930)	Flotation (single roughing and two cleaning stages; middlings returned to roughers and primary cleaners).
Old Dominion, Ariz., 1929 [18]	Principally chalcopyrite and pyrite with some chalcocite, bornite, and cuprite in quartzite and limestone.	[19] 2. 500 [8] 0. 109	Flotation (4 roughing stages—first 2 produce final concentrates. Concentrates from 3d stage and cleaned concentrate from 4th stage reground and returned to head of circuit).
Nevada Consolidated, Hayden, Ariz., 1928.[20]	Chalcocite with pyrite and secondary quartz in quartz-sericite schist gangue.	[2] 1. 250 [19] 1. 110	Flotation (primary and secondary classifier overflows conditioned separately and then combined and treated by two roughing stages. Final tailing from second rougher. Primary rougher concentrate cleaned once.)
Sherritt-Gordon, Manitoba, 1931.[21]	Chalcopyrite, pyrrhotite, pyrite. Massive sulfide in pegmatite and gneiss.	3. 67	Flotation (16-cell machines; first cell produces finished concentrate; concentrate from next 12 cells is double-cleaned; from last 3 cells reground and returned to head of circuit).
New Cornelia, Ariz., 1928 [22]	Malachite with some oxides and silicates of copper disseminated in monzonite porphyry.	[2] 1. 13	Leaching in tanks with dilute solution of sulfuric acid and ferric sulfate, clarification of solution, reduction of ferric iron with SO_2 gas, precipitation of copper by electrolysis.
New Cornelia 1 month 1938.[23]	Chalcopyrite with subordinate bornite, chalcocite, and native copper in porphyritic quartz monzonite.	[2] 0. 97 [6] 0. 01	Flotation (roughing producing concentrate and final tailing. Rougher concentrate cleaned once to produce final concentrate and a middling product. Middling reground and cleaned in two stages.)

TABLE 59.—*Data on copper-ore concentrators*

1. GENERAL DATA

Mine, reference, and year	General nature of ore treated; copper and associated minerals and rocks	Mill heads, copper, percent	General method of treatment
Calumet & Hecla Conglomerate, Mich., 1929.[1]	Native copper in hard conglomerate (felsite, porphyry, and quartz-porphyry pebbles).	[2] 2.223	Picking, jigging, and tabling followed by regrinding and tabling, leaching of sand tailing, and flotation of slime tailing.
Harmony, Idaho, 1929 [3]	Chalcopyrite and bornite intimately associated with pyrite in quartz gangue.	[2] 4.59	Flotation (1-stage roughing followed by 1-stage cleaning).
Cananea, Mexico, 6 months 1929.[4]	Bornite, chalcopyrite, tetrahedrite, pyrite in silicified lime and garnet gangue, and chalcocite in soft feldspathic gangue.	3.29	Differential flotation (3 roughing stages, primary concentrate cleaned twice, secondary and scavenger concentrate returned with cleaner middling to head of circuit).
Magma, Ariz., 1929 [5]	Bornite, chalcopyrite, and pyrite carrying gold and silver in silicified diabase with a quartzite gangue. Mineralization comparatively coarse.	[2] 7.12 [6] .04	Rough concentration on tables; tailing reground and treated by 2 roughing flotation stages (secondary rougher concentrate or middling finished by single cleaning. Rougher and cleaner both make finished concentrate and tailing).
United Verde, Ariz., Sept. 1929.[7]	Copper, iron, and zinc sulfide replacement in schist. Copper chiefly as chalcopyrite.	[2] 3.06 [6] .034 [8] .170	Flotation (first 4 cells produce final concentrate; secondary concentrate or middling from last 12 cells returned to head of circuit).
Nacozari, Mexico, last half 1929.[9]	Chalcopyrite and pyrite with small amounts of chalcocite, bornite, and covellite in andesite and latite breccia.	[2] 2.71 [6] .05	Flotation (2 roughing stages. Primary rougher concentrate cleaned in 2 stages; secondary rougher concentrate or middling returned to head of circuit).
Copper Queen, Ariz., 1929 [10]	Finely disseminated chalcocite associated with pyrite in siliceous porphyry gangue.	[2] 2.15 [8] .13	Flotation (overflows from primary and secondary classifiers treated separately by 1-stage flotation).
Morenci, Ariz., 1929 [11]	Chalcocite and copper oxides, carbonates and silicates finely disseminated in quartz monzonite porphyry.	[2] 1.916 [8] .225	Primary tabling followed by two stages of flotation treatment at 48- and 65-mesh sizes, respectively.
Verde Central, Ariz., 1929–30.[12]	Massive and disseminated chalcopyrite and pyrite with quartz.	2.67	Flotation (2 roughing stages; both concentrates combined and cleaned once, producing finished concentrate and middling that is returned to head of roughers).

[1] Benedict, C. H., Milling Method and Cost at the Conglomerate Mill of the Calumet & Hecla Consolidated Copper Co.: Bureau of Mines Inf. Circ. 6354, 1930, 22 pp.

[2] Total copper.

[3] Gardner, R. D., Milling Methods and Costs at the Harmony Mines, Baker, Idaho: Bureau of Mines Inf. Circ. 6285, 1930, 18 pp.

[4] Tye, A. T., Milling Methods and Costs at the Concentrator of the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico: Bureau of Mines Inf. Circ. 6261, 1930, 21 pp.

[5] Rose, J. H., and McNabb, J. C., Milling Methods and Costs at the Concentrator of the Magma Copper Co., Superior, Ariz.: Bureau of Mines Inf. Circ. 6319, 1930, 20 pp.

[6] Oxide copper.

[7] Kuzell, C. R., and Barker, L. M., Milling Methods and Costs at the Concentrator of the United Verde Copper Co., Clarksdale, Ariz.: Bureau of Mines Inf. Circ. 6343, 1930, 28 pp.

[8] Acid-soluble copper.

[9] Rose, E. H., and Cramer, W. B., Milling Methods and Costs at the Nacozari Concentrator of the Phelps Dodge Corporation, Nacozari, Sonora, Mexico: Bureau of Mines Inf. Circ. 6358, 1930, 38 pp.

[10] Wittenau, E., and Cramer, W. B., Milling Methods and Costs at the Copper Queen Concentrator of the Phelps Dodge Corporation, Bisbee, Ariz.: Bureau of Mines Inf. Circ. 6404, 1931, 29 pp.

[11] Crowfoot, Arthur, Concentrating Methods and Costs at the Morenci Concentrator of the Phelps Dodge Corporation, Morenci, Greenlee County, Ariz.: Bureau of Mines Inf. Circ. 6460, 1931, 36 pp.

[12] Dickson, R. H., and Smith, E. M., Milling Methods and Costs at the Verde Central Concentrator, Jerome, Ariz.: Bureau of Mines Inf. Circ. 6489, 1931, 12 pp.

Mine	Primary crushers				Secondary crushers					Capacity of crushing plant, tons per hour
	No.	Type	Size	Size of product, inches	No.	Type	Size	Size of product		
								Percent	Size, inch or mesh	
Calumet & Hecla Conglomerate, Mich.	1	Jaw	24 x 28 in	?	?	Steam stamps	Moving parts, 6,000 lb.	14. 5 28. 3	+10-mesh −200-mesh	Each jaw crusher, 200
Harmony, Idaho	1	do	16 x 18 in	−1½	1	Rolls	30 x 18 in	30 10	−2+1 inch +200−70-mesh	25
Cananea, Mexico	1	Gyratory	No. 8, type N, Allis Chalmers.	−3½	2	Cone	48 in	25. 9 49. 4 6. 8	−2+1 in −1+½ inch −8+10-mesh	220
Magma, Ariz	1	Jaw	12 x 24 in	−3½	1	do	do	Set at ⅝ in		100
United Verde, Ariz	4	do	48 x 36 in	−4	3 3	Disk followed by— Rolls	do 56 x 24 in		−1 in −¼ in	
Nacozari, Mexico, last half 1929.	2 1	Gyratory do	No. 8 McCully No. 4 Telsmith	−4 −1½	2	Disk	48 in	4. 42 6. 96	+1½ in −200-mesh	190
Copper Queen, Ariz	1 2	Jaw Gyratory	84 x 66 in No. 8 McCully	−10 −4	4	do	do	7. 1 13. 4	+1¼ in −200-mesh	400
Morenci, Ariz	1 [24] 2	Jaw Cone	36 x 48 in 7 ft	6.3 percent+ 0.742 in. 90.3+100-mesh	4	Rolls in closed circuit with secondary screens.	54 x 24 in	67. 4 19. 0	+48-mesh −200-mesh	335
Verde Central, Ariz	1 1	Jaw Cone	12 x 24 in 36 in	−2½ −⅝	1	Rolls	56 x 24 in		−¼ in	25–27
Matahambre, Cuba	2	Gyratory	No. 6 Kennedy	−4½	1 1	Cone Rolls in closed circuit with screens.	48 in 54 x 20 in	19. 5 33. 7 [25] 1. 5 26. 6	+1 in −⅜ in +½ in −10-mesh	100

[2] Total copper.

[6] Oxide Copper.

[8] Acid-soluble copper.

[13] Kirchner, A. R., Galloway, J. V., and Schoder, W. P., Milling Methods and Costs of the Minas de Matahambre, S. A., Concentrator: Bureau of Mines Inf. Circ. 6544, 1931, 11 pp.

[14] Nelson, W. I., Milling Methods and Costs at the Superior Concentrator of the Engels Copper Mining Co., Plumas County, Calif.: Bureau of Mines Inf. Circ. 6550, 1932, 22 pp.

[15] Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Bureau of Mines Inf. Circ. 6573, 1932, 25 pp.

[16] Munro, A. C., and Pearse, H. A., Milling Methods and Costs at the Concentrator of the Britannia Mining & Smelting Co. (Ltd.), Britannia Beach, B. C.: Bureau of Mines Inf. Circ. 6619, 1932, 24 pp.

[17] Martin, H. S., Milling Methods and Costs at the Arthur and Magna Concentrators of the Utah Copper Co.: Bureau of Mines Inf. Circ. 6479, 1931, 25 pp.

[18] Forrester, D. L., and Cramer, W. B., Milling Methods and Costs at the Concentrator of the Old Dominion Co., Globe, Ariz.: Bureau of Mines Inf. Circ. 6467, 1931, 30 pp.

[19] Sulfide copper.

[20] Garms, W. I., Concentrator Methods and Costs at the Hayden Plant of the Nevada Consolidated Copper Co., Arizona: Bureau of Mines Inf. Circ. 6241, 1930, 25 pp.

[21] Staff, Sherritt-Gordon Mines (Ltd.), History, Development, and Production Plans at Sherritt-Gordon Mines: Trans. Canadian Inst. Min. and Met., vol. 53, March 1932, p. 104. Chapman, Thomas G., Concentration of Copper Ores in North America: Bureau of Mines Bull. 392, 1936, 169 pp.

[22] Bell, G. A., Leaching Practice and Costs at the New Cornelia Mine of the Calumet and Arizona Mining Co., Ajo, Ariz.: Bureau of Mines Inf. Circ. 6303, 1930, 29 pp.

[23] Barker, L. M., Milling Methods at the New Cornelia Branch of the Phelps Dodge Corporation: Am. Inst. Min. and Met. Eng. Regional Meeting at Tucson, Ariz., Nov. 1, 1938.

[24] 1 is a spare.

[25] Screen undersize.

TABLE 59.—*Data on copper-ore concentrators*—Continued

2. CRUSHING PLANT—continued

Mine	Primary crushers				Secondary crushers					Capacity of crushing plant, tons per hour
	No.	Type	Size	Size of product, inches	No.	Type	Size	Size of product		
								Percent	Size, inch or mesh	
Engels, Calif.	1	do	Traylor No. 8	−3	2	Gyratory	No. 5 McCully, No. 6 Gates.	24	+2 in	100
	1	Jaw	24 x 36 in	−3	1	Rolls	54 x 24 in	4.23 4.24	+1.05 in −200-mesh	
Miami, Ariz.	3	Cone	7 ft	−1	3	do	77 x 24 in		−⅝ in	
					[26] 3	do	80 x 24 in		−¼ in	
Britannia, British Columbia.	1	Jaw	36 x 48 in	−6	2	Cone	5½ ft	30 56	+½ in + 4-mesh	350
	1	do	24 x 36 in		[26] 4	Rolls	54 x 18 in	3.5	+4-mesh	
					[26] 1	do	72 x 20 in	33.5	−65+100-mesh	
Utah Copper, Utah:										
(1) Magna	1	Gyratory	No. 27	−6	4	Cone	7 ft	23.16 6.93	+.742 in −200-mesh	1,450±
(2) Arthur	1	do	do	−6	4	do	do			
(1) Magna	1	do	do	−6	[26] 36	Rolls [27]	44 x 16 in	.52	+10-mesh	
(2) Arthur	1	do	do	−6	[26] 26	do.[27]	do	26.16	−200-mesh	
Old Dominion, Ariz.: Unit No. 1	2	do	No. 7½ Kennedy	−4	1	Disk	48 in		−1½ in	125
					1	Rolls	60 x 24 in	13.4 9.8	+.525 in −200-mesh	
Unit No. 2	2	do	do	−4	1	Disk	48 in		−1½ in	125
					2	do	36 in	4.5 12.6	+.752 in −200-mesh	
					[26] 2	Rolls	43 x 16 in	4.7 9.2	+3-mesh −200-mesh	50–55
Nevada Consolidated, Hayden, Ariz.	4	do	No. 8 Gates	−3	4	do	72 x 20 in	27.92 11.90	+.525 in −200-mesh	
					[26] 16	do	42 x 16 in	1.1 43.4	+10-mesh −200-mesh	435
Sherritt-Gordon, Manitoba.	2	Jaw	30 x 42 in	−4½	1	Cone	7 ft	10.6 4.2	+⅜ in −200-mesh	125
					[26] 1	Rolls	72 x 24 in	5.6 2.2	+⅜ in −200-mesh	
New Cornelia, Ariz, 1928.	1	Gyratory	No. 24	−6	5	Disk	48 in		−1 in	
	4	do	No. 8	−3	10	do	do		−3-mesh	
New Cornelia, Ariz., 1938.	4	do	48 in	−6 to −8	4	Cone	7 ft	27.9 22.3	+1.05 in −4-mesh	1,300
	1	do	54 in		[26] 6	Short-head cones	do	1.3 31.7	+0.742 in −4-mesh	

3. GRINDING PLANT

Mine	Grinding mills						Circulating load, percent	New feed, tons per hour	Skeleton size-analysis			
					Grinding media				Feed		Product	
											Classifier overflow	
	No.	Kind	Size	Circuit	Kind	Size, inches			Percent	Inch or mesh	Percent	Mesh
Calumet & Hecla Conglomerate, Mich.	[28] 24	Pebble mill	8 x 6 ft	Open	Pebbles	4	0	4.5 per mill.	18.0 .5	+10-mesh +150 −100-mesh.	[29] 0.2 26.3	[29] +20 −200
Harmony, Idaho	1	Ball mill	7 ft. x 36 in	Closed with drag classifier [30] and cleaner middling.	Cast iron	2 and 3½	58	9	35.0 15.0	+4-mesh −70 +200-mesh	35.0 65.0	−70+200 −200
Cananea, Mexico	1	Rod mill [30]	6 x 12 ft	Closed with drag classifier.	Chrome-steel rods.	3¼ x 12 ft		108	4.7 33.2 8.7	+1 in. +½ in −200-mesh	4.66 32.16	+10 −200
	2	Ball mills [31]	8 ft. x 28 in	do	Chrome-steel balls.	2½			2.22 3.08	+10-mesh −200 in.	1.32 76.52	+48 −200
Magma, Ariz.	3	do. [30]	6 x 4½ ft	Closed with Callow screens.	do	5	40–50	30–34	1.7 53.7	+1 in −¾ in., +4-mesh.	Screen undersize, 2.7 20.9	+14 −200
	3	Tube mill [31]	5 x 10 ft	Closed with drag classifiers.	Cast-iron balls	2	100		2.7 1.5 5.3	−200-mesh +14-mesh −200-mesh	.5 57.4	+35 −200
United Verde, Ariz.	2	Ball mill [30]	8 x 12 ft	do	do	3¼	400	64	3.07 11.50	+3-mesh −200-mesh	2.3 56.3	+35 −200
	1	do. [31]	do	Closed with bowl classifier.	do	2½	100	64			.8 73.8	+65 −200
Nacozari, Mexico, last half 1929.	6	Rod mill [30]	6 x 12 ft	Open; followed by bowl classifier.		3½	0	105	4.42 6.96	+1½ in −200-mesh	.73 86.00	+48 −200
	6	do. [31]	do	Closed with drag classifier.		2½	170	105			.4 58.0	+35 −200
Copper Queen, Ariz.	[32] 8	do. [30]	6 x 12 ft	Open, followed by bowl classifiers.	High-carbon steel.	3	0	178	1.4 13.4	+20-mesh −200-mesh	Sand 2.0 12.8 overflow. 2.4 75.9	+20 −200 +65 −200
	8	Ball mill [31]	do	Closed with drag classifier.	Chilled semi-steel.	3½			.9 45.6	+20-mesh −200-mesh	1.0 73.5	+65 −200

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[26] Tertiary crushing.
[27] In closed circuit with impact screens.
[28] Regrind mills; only 12 used in 1929.
[29] Pebble-mill tailing.
[30] Primary.
[31] Secondary.
[32] In series with bowl classifier and ball mills.

TABLE 59.—*Data on copper-ore concentrators*—Continued

3. GRINDING PLANT—continued

Mine	Grinding mills						Circulating load, percent	New feed, tons per hour	Skeleton size-analysis			
					Grinding media				Feed		Product	
											Classifier overflow	
	No.	Kind	Size	Circuit	Kind	Size, inches			Percent	Inch or mesh	Percent	Mesh
Morenci, Ariz	3	Ball mill [30]	7 x 12 ft	Closed with drag classifiers.	Cast iron	3	125	200			7.0 53.0	+48 −200
	3	do.[30]	8½ x 12 ft									
	5	do.[31]	6½ x 12 ft	do	do	2					1.5 72.0	+65 −200
Verde Central, Ariz.	2	do	8 ft. x 36 in	do	do	3		16.3	100	−¼ in	4.0	+48
Matahambre, Cuba.	4	do.[30]	6 x 4½ ft	Closed with trommels	Steel	4½	110	54	1.5 26.6	+½ in −10-mesh	.6 25.3	+8 −200
	4	do.[31]	6 x 6 ft	Closed with drag classifiers.	do	3	150	15	.6 25.3	+8-mesh −200-mesh	7.0 54.5	+48 −200
	1	do.[31]	5 x 5 ft	do								
Engels, Calif	3	do.[30]	8 ft. x 36 in	Open	Cast steel	5	0	15.9	1.66 3.99	+1.05 in −200-mesh	.90 54.79	+48 −200
		do.[30]	8 x 6 ft	Partly closed with 1 secondary mill.	do	5	33	24.5	1.83 4.80	+1.05 in −200-mesh	.02 62.70	+35 −200
	1	Tube mill [31]	6 x 12 ft	Closed	do	2½	315	12.05	3.71 10.69	+8-mesh −200-mesh	.06 65.35	+48 −200
	1	do.[31]	do	Closed with rakes [32]	do	2½	340	9.05	2.63 13.03	+3-mesh −200-mesh	.16 65.32	+48 −200
	1	do.[31]	6 x 10 ft	do	do	2½	340					
	1	Ball mill [31]	8 ft. x 36 in	do	do	2½	340					
Miami, Ariz	12	do.[30]	do	Partly closed with rakes								
	12	do.[31]	do	Closed with rakes								
	6	do.[33]	10½ x 8 ft	Closed with bowl classifier.								
	2	do.[34]	6 x 12 ft	do								
	1	do.[35]	6 ft. x 22 in	do								
Britannia, British Columbia.	6	do.[30]	6 x 11 ft	Open	Scrap steel	3½	0		3.5 33.5	+4-mesh −65+100-mesh.		
	18	do.[31]	5½ x 9 ft	Closed with drag classifier.	do	3½	150–200					

Utah Copper, Utah:												
(1) Magna	18	do.[30]	7 x 10 ft	Closed with drags	Scrap iron and steel and ferrochrome.	2⅜		1,170 (Apr. 1930)	.52 26.16	+10-mesh −200-mesh		
	36	do.[31]	do	do	do	2⅜					9.60 68.49	+100 −200
(2) Arthur	13	do.[30]	do	Open	do	2⅜			.52 26.16	+10-mesh −200-mesh		
	39	do.[31]	do	Closed with drags	do	2⅜					9.60 68.49	+100 −200
Old Dominion, Ariz.:												
Old plant	4	do.[30]	6 ft. x 50 in.	Open	Cast iron	2½			4.7 9.2	+3-mesh −200-mesh	25.8 35.9	+35 −10 −200
	1	do.[31]	do	Closed with drags	do	2	360	50–55	25.8	+35-mesh	1.2	+48
	1	do.[31]	8 ft. x 36 in.	do	do	2	360		35.9	−200-mesh	74.7	−200
New plant	1	Rod mill	6 x 12 ft	do	Steel	3 in. x 12 ft.	300	10–12	2.3 11.2	+.725 in. −200-mesh	5.5 52.2	−35 +48 −200
Nevada Consolidated Hayden, Utah, 1928.	16	Ball mill	7 x 10 ft	Closed with bowl classifiers.	Scrap steel	2	264	368	1.1 43.4	+10-mesh −200-mesh	2.2 72.5	+65 −200
Sherritt-Gordon, Manitoba.	[30] 3	do	8 ft. x 60 in.	Partly closed with rake classifiers.	Forged steel	3	"Light"	38.8	9.9 7.2	+3/16 in. −200-mesh	22.4 26.6	+35 −200
	[31] 3	do	8 ft. x 72 in.	do	do	2	"Light"		22.4 26.6	+35-mesh −200-mesh	3.6 43.2	+35 −200
New Cornelia, Ariz., 1928.	None											
New Cornelia, Ariz., 1938.	[30] 16	Ball mill	6 x 15 ft	Partly closed with spiral classifiers.	Cast semisteel	3	200±	[36] 833 [37] 572	25.6 6.7	+4-mesh −200-mesh	[38] 2.6 [38] 17.4	[38] +4 [38] −200
	[31] 16	do	do	Closed with spiral classifiers.	do	2½	500±				1.3 55.3	[39] +48 [39] −200
	[34] 1	do	8 x 12 ft	Closed with bowl classifiers.								

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[30] Primary.
[31] Secondary.
[32] In series with bowl classifier and ball mills.
[33] Tertiary mills.
[34] Concentrate regrind.
[35] Middling regrind.
[36] Capacity.
[37] During period covered by metallurgical and cost data.
[38] Primary mill discharge.
[39] Classifier overflow.

TABLE 59.—*Data on copper-ore concentrators*—Continued

4. CONCENTRATION, FIRST STAGE

Mine	Method	Feed						Products				
		Source	Tons per hour	Copper, percent	Solids, percent	Skeleton size		Concentrate		Middling	Tailing	
						Per cent	Mesh	Tons per hour	Copper, percent		Tons per hour	Copper, percent
Calumet & Hecla Conglomerate, Michigan.	Jigging	Stamp-mill screen discharge.		2.95		14.5 28.3	+10 −200		39.65 final.	Hutch to tables	Jig overflow to flotation.	
Harmony, Idaho	Flotation roughing.	Classifier overflow	11	4.59	55.9	10.0 65.0	−70, +100 −200	3.2[40]	15.7	None	7.8	0.02 final.
Cananea, Mexico	do	do	100	2.44		.38 73.16	+48 −200		23.42	To cleaner		.15.
Magma, Ariz	Shaking tables	Screen undersize	27	7.12		2.7 20.9	+14 −200	4.8	15.92 final.	Return to Callow screens.	22.2	5.23 to tube-mill classifier.
United Verde, Ariz	Flotation	Bowl classifier, overflow.	64	3.06	25.6	.8 73.8	+65 −200	12.5	14.42	Return to head of cells.	51.5	.32.
Nacozari, Mexico, last half 1929.	(1) Flotation cleaning slime section.	do	[41] 32.8 [42] 39.3	2.93 3.15	19 average all sections.	.73 86.00	+48 −200	4.1 final	22.47		To roughers.	
	(2) Sand section	Drag classifier, overflow.	[41] 71.3 [42] 84.7	2.61 2.70		.40 58.00	+35 −200	To cleaners			To secondary roughers.	
Copper Queen, Ariz	(1) Slime section	Bowl classifier, overflow.	80–90	1.60	22	2.4 75.9	+65 −200		13.54 final.	Returned to head of cells.		0.22 final.
	(2) Sand section	Drag classifier, overflow.	90–100	2.48	23–25	1.0 73.5	+65 −200		11.30 final.	do		.18 final.
Morenci, Ariz	Shaking tables	Screen undersize	205	1.82		67.4 19.0	+48 −200	4.0	19.59 final.	To primary grinding.	None.	
Verde Central, Ariz., May and June 1930.	(1) Flotation roughing.	Classifier overflow	16.3	2.67		3.5 51.0	+48 −200		15.0 to cleaners.			0.6 to secondary roughers.
	(2) do	Primary rougher tailing.										.20 final.

Matahambre, Cuba	Flotation (primary roughers).	Secondary ball-mill classifier overflow.	54	4.55	35.8	7.0 54.5	48 −200		19.7–28.4	To secondary roughers.		
Engels, Calif	Flotation (primary).	Classifier overflows	45.5	1.54	26–28	2.80 62.42	+65 −200	1.4 first cell final.	29.18	Cells 2–4 to finisher; cells 5–7 to primary cleaners; cells 8–12 to secondary cleaners.		0.16 (to scavenger).
Miami, Ariz	Bulk flotation	Bowl-classifier overflow.	714	[2].716 [19].604		4.23 50.88	+48 −200		To cleaner.	Recirculated to head of cells via bowl and regrind.		0.153.[2] 0.069 [19] final.
Britannia, British Columbia, July 1931.	Bulk flotation. (1) primary slime.	Drag classifier following grizzly and screens.				.9 58.4	+28 −200	32.0–40.0	10–11.0			0.16–.30 to scavenger cells.
	(2) Ground ore	Ball-mill classifier overflow.				3.0 39.5	+28 −200		9.0–10.0			0.07–.10 to scavenger cells.
Utah Copper, Utah:												
(1) Magna	Flotation roughing.	Thickened ball-mill classifier overflow.	1,170 (Apr. 1930)	.957	24–26	9.60 68.49	+100 −200		? To cleaners.	Returned to head of roughers.		
(2) Arthur	do	do			27–30	68.49	−200		do	do		0.1046.[2] 0.0746.[19]
Old Dominion, Ariz	Flotation roughing, first two stages.	Classifier overflow	[43] 55.6 [44] 63.0	[2] [43] 2.61 [19] [43] 2.501 [44] 2.53	22	[44] 6.2 55.3	[44] −48 −200	6.9 [43] 5.4 [44]	18.97 [43] 25.22 [44]	To tertiary flotation.		To scavenger or final rougher.
Nevada Consolidated (Hayden), Ariz. 1928.	Flotation; primary roughing.	do	368	1.250	32	2.2 72.5	+65 −200		To cleaner cells.	To secondary rougher.		
Sherritt-Gordon, Manitoba, June to November 1931.	Flotation roughing 1st cell.	do	38.8	3.67	35	3.6 43.2	+35 −200	First cell final.	22.79	To cells 2–13		
	Cells 2–13	First cell								To cells 14–16		
	Cells 14–16	13th cell								To cleaner		
New Cornelia, Ariz., 1938.	Flotation roughing.	Classifier overflow	572	[2].97	20–24	1.3 55.3	+48 −200	23 to 36 to bowl classifier.	16.0–25.0		550	To waste.

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[2] Total copper.
[19] Sulfide copper.
[40] 15 percent final concentrate; balance to cleaner cells.
[41] New feed.
[42] Including return feed.
[43] 1929.
[44] 1931.

TABLE 59.—*Data on copper-ore concentrators*—Continued

5. CONCENTRATION, SECOND STAGE

Mine	Method	Feed					Products					
		Source	Tons per hour	Copper, percent	Skeleton size		Concentrate			Tailing		
					Per cent	Mesh	Tons per hour	Copper, percent	Disposition	Tons per hour	Copper, percent	Disposition
Calumet & Hecla Conglomerate, Mich.	(1) Tabling	Jig hutch							Final concentrate		0.70	Regrind mill.
	(2) Tabling	Jig overflow sands							do			
Cananea, Mexico	Flotation cleaning.	Rougher middling		0.93	0.08 91.54	+40 −200		5.36	To recleaner	Middling	.68	To rod-mill classifier.
Magma, Ariz	Flotation roughing.	Tube-mill classifier overflow.	22.1	5.23	.5 57.4	+35 −200	6.3	17.97 final	To dewatering	Middling Tailing	2.0–3.0 .198	To cleaner. To waste.
Nacozari, Mexico, last half 1929.	Flotation secondary rougher sand section.	Primary rougher tailing.							Returned to drag classifier.	64	.13	Final to waste.
Morenci, Ariz	Flotation (primary roughing).	Primary ball mills table middling.	200	1.56	7.0 53.0	+48 −200			To cleaner cells		.53	To secondary grinding.
	Flotation (secondary roughing).	Primary tailing and cleaner middling after secondary grinding.			1.5 72.0	+65 −200					[45].358	To waste.
Matahambre, Cuba, 1930.	Flotation, (2) secondary roughers.	Primary middling						14.3–15.2	To head of circuit		1.05–1.40	To tertiary rou ers.
	(3) Tertiary roughers.	Secondary middling.						9.1	do		.27– .30	To scavengers
	(4) Scavengers	Tertiary tails						5.2	do		.216	Final tailin waste.
Engels, Calif., 1929	Flotation finishing.	Primary cells 2–4	1.65	13.42			.755	27.0	Final	2.21	12.18	To primary er.
	Primary cleaning	Primary cells 5–7	1.81	3.13			1.31	19.98	To finisher cells	3.18	3.37	To secc cleaner.

Miami, Ariz., 1930	Bulk rougher concentrate cleaning.	Bulk flotation concentrate.	------	------	------	------	------	15.74	To selective flotation.	------	------	Returned to head of bulk flotation.
Britannia, British Columbia.	Scavenging cells (roughing and cleaning).	Bulk tailing	------	------	------	------	------	------	To concentrate regrind.	------	.11	To waste.
Utah Copper, Utah (1) Magna.	Primary flotation cleaning.	Rougher concentrate.	------	------	------	------	------	------	To second cleaners.	------	------	To head of roughers.
Old Dominion, Ariz.	Flotation roughing, 3d stage.	From second stage	------	------	------	------	------	10.0–12.0	To regrinding mills	------	------	Scavenger flotation.
Nevada Consolidated (Hayden), Utah.	Flotation roughing, 2d stage.	Middling from primary.	------	------	------	------	------	------	To head of primary roughers.	------	[2].171 [19].079	To waste.
Sherritt - Gordon, Manitoba.	Flotation roughing, cells 2–13.	From 1st cell	------	------	------	------	------	------	To 6th cell of cleaner.	------	------	To cell 14.
	Cells 14–16	From cell 13	------	------	------	------	------	------	Joins tailing from cleaner.	------	[46].215	Final.
New Cornelia, Ariz., 1938.	Flotation cleaning, primary.	Rougher concentrate.	23–36	16.0–25.0	------	------	------	31.16	Final	------	------	Middling to bowl classifier. Sands reground and join overflow to 2-stage cleaning.

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[2] Total copper.
[19] Sulfide copper.
[45] 0.205 percent acid soluble; 0.153 percent sulfide.
[46] Computed.

TABLE 59.—*Data on copper-ore concentrators*—Continued

6. CONCENTRATION, FINAL

Mine	Feed					Products					Entire plant	
	Method	Source	Copper, percent	Skeleton size		Concentrates		Tailings		Recovery, percent	Ratio of concentration	Over-all recovery, percent
				Per cent	Mesh	Tons per hour	Copper, percent	Tons per hour	Copper, per cent			
Calumet & Hecla Conglomerate, Mich.	(3) Flotation	Table-tail and other slimes.	0.580	0.05 80.0	+48 −200	1.08	35.05	95	0.109	[47] 81.2	21.6–1	95.
	(4) Ammonia leaching.	Table-tail sands	.736	2.0 8.0	+28 −200				.108	[48] 85.3		
Harmony, Idaho	Second stage is final					1.2	28.61				7.5–1	99.2.
Cananea, Mexico	Flotation recleaning	First cleaner	5.36	.10 82.42	+48 −200		12.66	Middling to rod mill classifier.	2.37		6.8–1	95.74.
Magma, Ariz.	Flotation cleaner	Rougher middling	2.0–3.0				17.97		.198		2.43–1	98.37
United Verde, Ariz.	(49)	(49)				12.5			.32		5.1–1	91.13.
Nacozari, Mexico, last half 1929.	Flotation, (1) slime section roughers.	Cleaner tailing				4.1	22.47	28.2	.11	63.28	8.9–1	95.86.
	(2) Sand section cleaners, 2 stages.	Primary rougher concentrate.				7.6	23.67	(50)	(50)	32.58		
Copper Queen, Ariz.	(51)	(51)	(51)	(51)	(51)	27.17	12.92	150.8	.22	91.32	6.58–1	91.32. [19] 94.58.
Morenci, Ariz.	Flotation cleaning [52]	Primary concentrate Secondary concentrates.					19.80	Returned to secondary grinding.		76.49	14.81–1	82.56.
Verde Central, Ariz., May and June 1930.	do	Rougher concentrate	15.0			2.3	19.83	Returned to roughers			7.95–1	93.2.
Matahambre, Cuba	Flotation cleaning, 2 stages.	Primary rougher concentrate.	19.7–28.4			19.4	29.20	do			6.7–1	96.0.
Engels, Calif.	Secondary cleaner	Primary cells 8–12 and primary cleaner.	1.92			.47	11.22 (to primary cleaner).	4.15 (to head of circuit.)	1.96			
	Scavenger cells	Primary tailing	.16			6.95	1.05 (to head of circuit).	4.33	.15		19.94–1	91.34.

Miami, Ariz	Differential flotation, 3 stages of roughing and 1 of cleaning.	Bulk concentrate regrind mills.	14.511	3.6 45.8	+48 −200	11.82	34.127	First stage tailing to No. 1 scavengers; from No. 1 scavengers to No. 2 scavengers; from No. 2 scavengers to head of roughers.		60.4–1	78.88, gross 88.76.[19]
Britannia, British Columbia, July 1931.	Differential flotation, 2 stages of cleaning.	Bulk concentrate, scavenger cleaner concentrate.	10.0–11.0	3.0 76.0	+100 −200	8.74	24.0–26.0 (roughing); 27.73 (final cleaning).	3.0–4.0 (to scavengers)		30.9–1	89.36. [19] 90.71.
Utah Copper, Utah, Magna, April 1930.	Secondary flotation cleaning.	Primary cleaner concentrate.				31.8		To head of primary cleaners.		36.805–1	89.37 (total copper).
Old Dominion, Ariz	Scavenger flotation	Tailing from tertiary cells.					1.0–2.0 to regrinding mill.	Final: [19] 0.203 (1929) [19] .190 (1930)		8.10–1 (1929) 12.36–1 (1930)	[19] 93.84. [19] 93.33.
Nevada Consolidated (Hayden), Ariz., 1928.	Flotation concentrate cleaning.	Primary rougher concentrate.				20	[19] 19.248 [2] 20.246	To head of primary roughers.		18.61–1	87.06.
Sherritt-Gordon, Manitoba.	Flotation cleaning	Middling from primary cells 2–13.				First 2 cells final; next 4 cells to 2d cell or final; last 10 cells to 2d cell.		Join rougher concentrates, cells 14–16, and returned to regrind.		6.5–1	95.07.
New Cornelia, Ariz., 1928.[36]	(53)	(53)	(53)	(53)	(53)	(53)		(53)	(53)	(53)	75.02.
New Cornelia, Ariz., 1938.	Two-stage flotation cleaning.	Primary cleaning middling.				Second stage final, 31.16.		First stage to scavenger cells; second stage back to first stage.			[2] 92.46. [19] 93.40.
	Scavenging	First-stage middling				To head of two-stage cleaning.		Final		34.84–1	Silver, 84.55; gold, 80.34.

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[2] Total copper.
[19] Sulfide copper.
[36] Capacity.
[47] Percent recovery of copper in flotation feed.
[48] Percent recovery of copper in leaching-plant feed.
[49] See (4) first stage.
[50] Returned to drag classifiers.
[51] See first stage, which is final.
[52] 1 stage on primary rougher concentrate and 2 stages on secondary rougher concentrate.
[53] Leaching and precipitation by electrolysis.

TABLE 59.—*Data on copper-ore concentrators*—Continued

7. DEWATERING, FILTERING, TAILING DISPOSAL

Mine	Concentrates						Tailings
			Dewatering		Filtering		
	Source	Moisture, percent	Method	Moisture in product, percent	Method	Moisture in cake, percent	Method of disposal
Calumet & Hecla Conglomerate, Mich.	Picked, jig, and table concentrates.		Drainage in bins	7.0		7.0	
	Flotation		Thickeners		Drum filters	14.0	Pumped to lake.
Cananea, Mexico	Combined primary and recleaner concentrates.	82–85	Thickeners; overflow to head of mill.	30.0	do	9.8	Laundered to tailing dam; water pumped back to mill tanks.
Magma, Ariz	Tables and flotation		2 stages agitator-thickeners.[54]	30–35 [54]	Filter plant at smelter		Do.
United Verde, Ariz	Flotation	60–65	2-stage thickening	25–30	Leaf filters	12–14	Thickened and run through wood-stave gravity pipeline to pond.
Nacozari, Mexico	Sand concentrate	50–60	Direct to filter	Direct to filter	Drum filter	10	Laundered ¾ mile to pond.
	Slime section concentrate.		Thickener	40		10	
Copper Queen, Ariz	Combined concentrates		Drag classifier	Coarse concentrate to tanks for drainage.	Drained in Blaisdall tanks.	6.5	Thickened and laundered to tailing dam.
				Fine concentrate to thickener.	Drum filter	11.33	
Morenci, Ariz	Table concentrate		2 drag classifiers and 1 vacuum classifier.	6–7	None		Dewatered in thickeners and water reclaimed; then laundered to tailing dam.
	Flotation concentrate		3 series thickeners, water reclaimed.		Drum filters		
Verde Central, Ariz	Cleaner concentrate	70	Thickener followed by rectangular tank.	30–50	Disk filter	11.0	Thickened to 35 percent solids, laundered 2 miles; overflow reclaimed.
Matahambre, Cuba	do	30	Drag classifier, oversize to filters, undersize to thickeners.		Drum filters		Thickened; pulp to bowl classifier; sands to mine for filling, overflow returned to plant by launder.
Engels, Calif	First cell of primary unit and finisher concentrate.	37.5	Thickening	−25	do	10.0–11.0	Flumed 4,600 feet to rake classifiers; slime overflow to thickener. Thickener underflow flumed 4 miles. Overflow to creek.

Miami, Ariz	Differential flotation cleaner.		do	40–50	do	8.0	
Britannia, British Columbia.	do		do		Leaf filters	10.6	Laundered and piped to tidewater.
Utah Copper, Utah, Magna (April 1930).	Secondary cleaners	75	Dorr tanks	40±	Disk filters	9.924	Run by gravity to pond of 10 square miles area.
Old Dominion, Ariz	First two flotation stages.		Spiral classifier	30 (sand) drains to 7.5 percent.			Rake classifiers at dam. Sands used to build dam.
		88	Thickener	88 (overflow), 40	Drum filters	16.23	
Nevada Consolidated (Hayden), Ariz.	Cleaner cells	80–85	Cone tank; underflow to drain bins.	−10	Underflow from first Dorr tank to drum filters.	−12.0	Laundered to 1,000-acre pond held by dikes on 3 sides.
		90–93	Overflow to Dorr tanks in series.	70–80			
Sherritt-Gordon, Manitoba.	Final concentrate		Genter thickener	40	Leaf filter	−8.0	
New Cornelia, Ariz., 1938.	do		Thickeners		4-drum; 2-disk	7.5	Thickened in traction thickeners discharging pulp at 45 percent solids. Water returned to mill.

8. OPERATING COSTS PER TON OF ORE MILLED

Mine and year	Ore treated per day, tons	Operating costs per ton													
		Crushing	Grinding, screening, classifying	Concentrating	Dewatering and filtering	Disposal of tailing	Other	Total	Labor	Power and water supply	Supplies			Other supplies and general	Total
											Crushing	Grinding	Reagents		
Calumet & Hecla Conglomerate, Mich., 1929	2,900	[55] $0.192	$0.081	[56] $0.072 [57] .039 [58] .101 .212		$0.017	$0.031	$0.533							
Harmony, Idaho, 4 months, 1929	200	.074	.336	.085	$0.057	.010	.089	.651	$0.243	$0.120	$0.020	$0.176	$0.017	$0.075	$0.651
Cananea, Mexico, 6 months, 1929	2,400	.037	.191	.131	.015	.008	.235	.617	.119	.203	.005	.077	.055	.158	.617
Magma, Ariz., 6 months, 1929	650	.081	.297	.202	.005	.014	.335	.934	.257	.194	.014	.171	.062	.236	.934
United Verde, Ariz., Sept. 1929	1,532	.203	.187	.181	.034	.004	.127	.736	.180	.151		.065	.115	.022	.533 [59] .203 .736

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[54] Agitation and high moisture required for pumping to filter plant.
[55] Stamping.
[56] Gravity concentration.
[57] Flotation.
[58] Leaching.
[59] Crushing—labor, power, and supplies.

TABLE 59.—*Data on copper-ore concentrators*—Continued

8. OPERATING COSTS PER TON OF ORE MILLED—continued

Mine and year	Ore treated per day, tons	Operating costs per ton													
		Crushing	Grinding, screening, classifying	Concentrating	Dewatering and filtering	Disposal of tailing	Other	Total	Labor	Power and water supply	Supplies			Other supplies and general	Total
											Crushing	Grinding	Reagents		
Nacozari, Mexico, 1928–29 [60]	[60] 2,516	$0.061	$0.223	$0.105	$0.017	$0.028	$0.079	$0.513	$0.137	$0.138	$0.024	$0.105	$0.067	$0.042	$0.513
Copper Queen, Ariz., 1929	4,281	.059	.194	.094	--------	.022	.124	.493	.155	.121	.015	.085	.042	.075	.493
Morenci, Ariz.:															
1929	4,739	.109	.150	.115	--------	.012	.193	.579	.177	.207	[61] .141	--------	--------	.054	.579
April 1930	5,106	.107	.125	.106	--------	.011	.170	.519	.181	.179	[61] .123	--------	--------	.036	.519
Verde Central, Ariz., June 1, 1929 to Aug. 1, 1930	350±	.380	.370	.190	.050	.080	.190	1.260	.584	.317	.161	.121	.051	.023	1.257
Matahambre, Cuba, 1930	1,300	.130	.279	.154	.033	.019	.132	.747	--------	--------	--------	--------	--------	--------	--------
Engels, Calif., 1929	1,091	.113	.285	.093	.033	.058	.070	.652	.214	.158	.031	.124	.033	.092	.652
Miami, Ariz., 1930	17,141	.058	.091	.077	.006	.012	.053	.297	.082	.077	--------	[62] .078	.040	.020	.297
Britannia, British Columbia, July 1931	6,486	.054	.046	.028	.009	.040	--------	.177	.069	.025	.021	.021	.017	.024	.177
Utah Copper, Utah, 29 days in 1929 (both plants)	48,500	.065	.092	.091	.005	.014	.035	.302	.082	.110	.013	.030	.035	.032	.302
Old Dominion, Ariz., 1929	1,338	.150	.187	.140	.042	.025	.155	.699	.220	.202	--------	.063	.069	.145	.699
Nevada Consolidated (Hayden), Ariz., 1928	8,838	.111	.078	.094	.017	.014	.085	.399	.105	.104	.040	.029	.046	.075	.399
Sherritt - Gordon, Manitoba, 8 months, 1931	838	.108	.352	.274	.056	.014	.020	.824	.124	.179	.011	.210	.126	.174	.824
New Cornelia, Ariz., leaching, 1928	29,300±	.148	None	[58] .357	[63] .434	[64] .045	.210	1.194 [65] .068	--------	--------	--------	--------	--------	--------	--------
New Cornelia, Ariz., flotation, 1 month, 1938	13,723	.088	.098	.048	.009	.013	.052	.308	.088	.087	--------	[61] .116	--------	.017	.308

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[58] Leaching.

[60] Costs for 1928-29 tonnage for last half 1929.

[61] Total supplies.

[62] All supplies except reagents.

[63] Electrolytic expense.

[64] Cement copper expense.

[65] Per pound of copper.

LEAD AND ZINC ORES

Minerals of lead and zinc commonly are rather closely associated in nature; zinc-free lead ores and lead-free zinc ores are exceptional rather than common, although there are such.

Lead and zinc ores usually contain some silver, but those of the Mississippi Valley region are silver-free. Indeed, lead-zinc ores often are commercially valuable chiefly because of their silver (and sometimes gold) content. The silver and gold usually are intimately associated with the base metal minerals, are recovered with them in the smelter, and are finally separated in the refinery.

Because of this close association of zinc and lead ores, it is necessary to consider them together in discussing their concentration.

An earlier summary of practices and costs of concentrating lead and zinc ores was published by the Bureau of Mines in 1935.[93] Most of the data in the present bulletin were abstracted from the earlier publication.

The development of differential or selective flotation has had an important influence upon the economics of lead and zinc mining and milling. Higher recoveries and better separation of the lead and zinc minerals into different concentrate products than was possible by gravity concentration has made it profitable to mine and treat many ores that formerly could not be worked profitably. Zinc concentrates with a high pyrite content produced by gravity methods sometimes required flash roasting and magnetic separation of the iron from the roasted material to produce a zinc concentrate of the required grade. With flotation, however, it is possible to depress the iron sulfide and float the zinc sulfides. With straight gravity treatment high slime losses were often unavoidable, but high recoveries can be made by flotation.

Gravity concentration still retains a place in the flow sheet of some mills where it is employed mainly on the coarse product from primary or secondary crushing before fine grinding, either to recover quickly coarse ore particles or to make a quick reject of gangue minerals, thus reducing the amount of material that must pass to expensive fine grinding.

Screens, classifying cones, or kindred settling devices, jigs, and shaking tables comprise the essential gravity concentration equipment. Tables are used incidentally in many flotation plants to check the performance of the flotation machines or the fineness of grinding or for other purposes.

METHODS OF TREATMENT

Because of the wide variations in tenor or ore, nature of ore and associated gangue minerals, fineness and distribution of ore particles in ores from different localities there is no standard method of treatment, and flow sheets must be worked out for each particular ore.

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[93] Jackson, Chas. F., Knaebel, John B., and Wright, C. A., Lead and Zinc Mining and Milling in the United States; Current Practices and Costs: Bureau of Mines Bull. 381, 1930, pp. 146–194.

However, for the purpose of the present discussion, methods of treatment may be classified roughly as follows:

Gravity methods:
1. Jigging:
    - (*a*) Concentration of sized material.
    - (*b*) Concentration by roughing and cleaning.
2. Table concentration:
    - (*a*) Concentration of different classifier sand products on separate tables.
    - (*b*) Concentration by roughing and retreatment of concentrate and middling products.
3. Flotation:
    - (*a*) Following gravity concentration; flotation of primary slimes, of screen undersize from sizing ahead of gravity concentration, or of reground middlings or tailings from gravity concentration.
    - (*b*) Straight bulk flotation producing a combined lead-zinc concentrate (when either lead or zinc occurs in very small proportion to the other).
    - (*c*) Differential or selective flotation by which the ground mill feed is first treated in a lead circuit by roughing and cleaning in one or more stages and producing finished lead concentrate, followed by reconditioning and flotation of the lead-circuit tailing to recover the zinc in one or more stages of roughing and cleaning, the zinc-circuit tailing being final tailing.
    - (*d*) Flotation of oxidized lead and zinc minerals by the use of special reagents for sulphidizing or otherwise conditioning the pulp and for stage-addition in the process, usually involving a high reagent cost. (See Chief Consolidated, table 60.)

Various combinations of these methods of concentration are employed to suit different ores.

Figure 155 is the flow sheet of a concentrator employing gravity methods followed by flotation (typical of practice in the Tri-State district). Figure 156 is the flow sheet of a simple selective flotation plant in which lead and zinc concentrates are each cleaned once and the middling products are returned to the heads of their respective circuits for retreatment.

Table 60 presents data on a number of typical lead and zinc ore concentrators and summarizes the methods employed, metallurgical results obtained, and the operating costs.

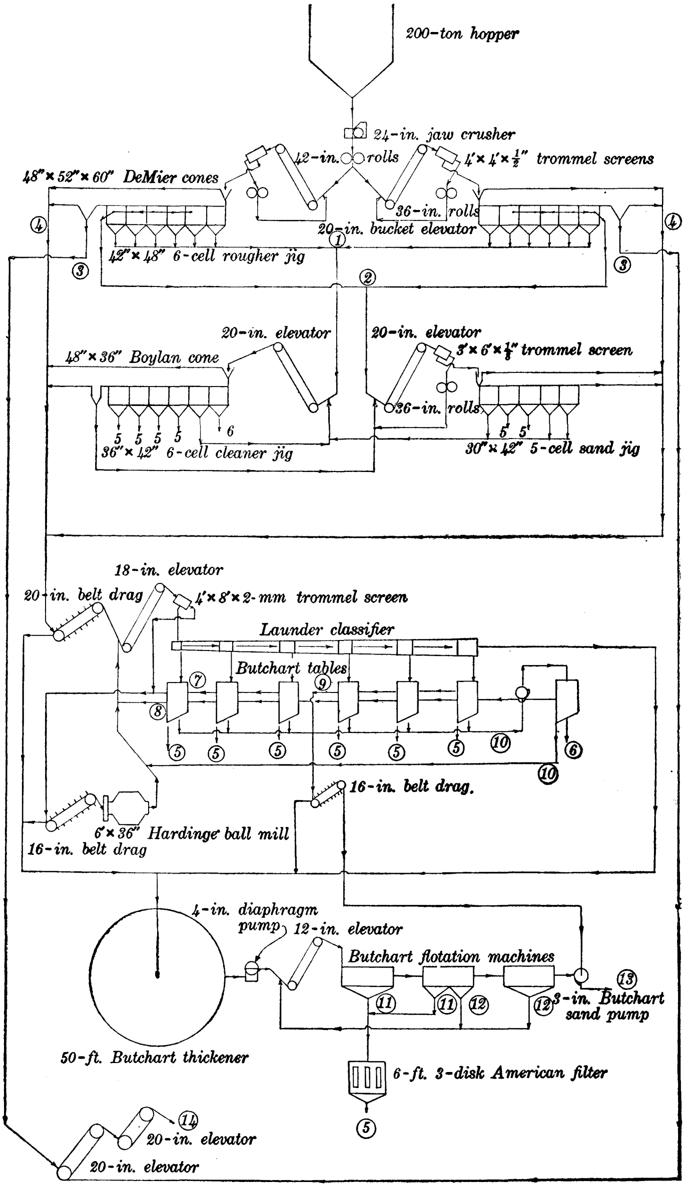
200-ton hopper
24-in. jaw crusher
42-in. rolls
4'×4'×½" trommel screens
48"×52"×60" DeMier cones
36-in. rolls
20-in. bucket elevator
42"×48" 6-cell rougher jig
①
②
③
④
20-in. elevator
20-in. elevator
48"×36" Boylan cone
3'×6'×⅛" trommel screen
36-in. rolls
5 5 5 5 6
36"×42" 6-cell cleaner jig
5 5
30"×42" 5-cell sand jig
18-in. elevator
20-in. belt drag
4'×8'×2-mm trommel screen
Launder classifier
Butchart tables
⑦
⑧
⑨
⑤
⑥
⑩
16-in. belt drag.
6'×36" Hardinge ball mill
16-in. belt drag
4-in. diaphragm pump
12-in. elevator
Butchart flotation machines
⑪
⑫
⑬
3-in. Butchart sand pump
50-ft. Butchart thickener
6-ft. 3-disk American filter
⑭
20-in. elevator
20-in. elevator

FIGURE 155.—Flow sheet of White Bird concentrator, Oklahoma; gravity followed by flotation.

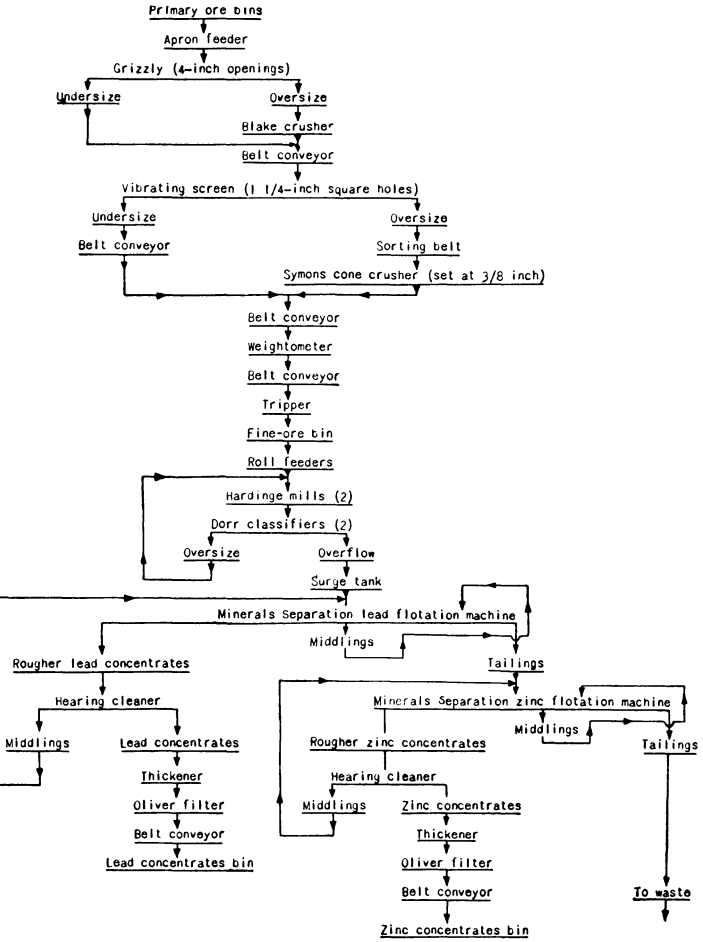
Primary ore bins
Apron feeder
Grizzly (4-inch openings)
Undersize
Oversize
Blake crusher
Belt conveyor
Vibrating screen (1 1/4-inch square holes)
Undersize
Oversize
Belt conveyor
Sorting belt
Symons cone crusher (set at 3/8 inch)
Belt conveyor
Weightometer
Belt conveyor
Tripper
Fine-ore bin
Roll feeders
Hardinge mills (2)
Dorr classifiers (2)
Oversize
Overflow
Surge tank
Minerals Separation lead flotation machine
Middlings
Rougher lead concentrates
Tailings
Hearing cleaner
Minerals Separation zinc flotation machine
Middlings
Lead concentrates
Middlings
Rougher zinc concentrates
Tailings
Thickener
Hearing cleaner
Oliver filter
Middlings
Zinc concentrates
Belt conveyor
Thickener
Lead concentrates bin
Oliver filter
Belt conveyor
To waste
Zinc concentrates bin

FIGURE 156.—Flow sheet of Tybo concentrator, Nevada (selective flotation).

TABLE 60.—*Data on lead and zinc ore concentrators*

1. GENERAL DATA

Mine and year	General nature of ore treated	Mill heads			General method of treatment
		Lead, percent	Zinc, percent	Silver, ounces per ton	
Mascot, Tenn., 1929 [1]	Sphalerite with some pyrite, quartz, and calcite in dolomite gangue.	0.0	2.9	0.0	Jigging by roughing and cleaning, discarding 50 percent as coarse tailing; [2] flotation of slimes and reground jig middling.
White Bird, Okla., 1929 (Aug., Sept., Oct.).[3]	Sphalerite and galena in chert and dolomite		8.99	.0	Jigging by roughing and cleaning, fine sands treated on tables. Flotation of slimes from cone and drag classifiers of gravity plant. Only zinc concentrates produced by flotation.
Black Hawk, N. Mex., Sept. and Oct. 1929.[4]	Sphalerite, galena, chalcopyrite, and pyrite in limestone and garnet gangue.	1.99	9.93	1.15	Some hand sorting of coarse waste followed by flotation (1 stage roughing and 2 stages of cleaning each for lead and zinc.)
Tybo, Nev., March 1930 [5]	Partly oxidized complex ore—galena and sphalerite containing gold and silver with pyrite in quartz and limestone gangue.	5.95	4.14	9.44	Flotation. (1 stage of roughing and 1 of cleaning in both lead and zinc circuits.)
Ruby, Ariz., 1929–30 (6 months).[6]	Mixed sulphides in quartz gangue; galena, sphalerite, and pyrite with minor chalcopyrite and tetrahedrite.	5.13	5.98	8.17 (0.07 oz. gold)	Flotation. (2 roughing stages producing finished concentrate in first stage and middlings that are re-treated in second stage in both lead and zinc sections.)
Morning, Idaho, 1930 [7]	Lead and zinc sulfides containing silver, with siderite and quartz gangue.	9.0	6.5	3.4	Flotation. (2 roughing stages, both lead and zinc, and 3 cleaning stages each.)
Hecla, Idaho, 1930 [8]	Principally argentiferous galena with some sphalerite and pyrite in crushed quartzite and siderite gangue.			4.77	Coarse ore sorted on belt, giving a crude lead shipping product and waste rock. Jigging of sized feed producing lead concentrate, followed by flotation in separate circuits of primary slime and jig tailing, producing lead concentrate, and selective flotation of reground fine-jig middlings and original −16 mesh feed.
Pecos, N. Mex., 1930 [9]	Massive sulfides—pyrite, sphalerite, galena, chalcopyrite, marmatite, chalcocite, bornite, argentite, and proustite in mica, quartz, tourmaline, and hornblende gangue.	4.93	15.41	3.37 (0.107 oz. gold)	Flotation. Micaceous gangue first removed by flotation, tailing going to lead section. Mica concentrate cleaned by flotation and tabling. Table concentrate goes to lead section and mica tailing to waste. Lead cells make rough and clean concentrate, middling (returned), and tailing to zinc section.

[1] Strachan, C. B., Milling Methods of the American Zinc Co. of Tennessee, Mascot, Tenn.: Bureau of Mines Inf. Circ. 6379, 1930, 12 pp.

[2] Later treatment includes use of differential density process to remove 45 percent of the gangue as tailing in the −1¼-inch, +½-inch size. (Eng. and Min. Jour., August 1938, pp. 49–54.)

[3] Crabtree, E. H., Jr., Milling Practice at the White Bird Concentrator, Canam Metals Corporation, Picher, Okla.: Bureau of Mines Inf. Circ. 6353, 1930, 10 pp.

[4] Wright, I. L., Milling Methods and Costs at the Black Hawk Concentrator, Hanover, N. Mex.: Bureau of Mines Inf. Circ. 6359, 1930, 15 pp.

[5] Blackburn, W. H., Milling Methods and Costs at the Concentrator of the Treadwell-Yukon Co. (Ltd.), at Tybo, Nev.: Bureau of Mines Inf. Circ. 6430, 1931, 14 pp.

[6] Andrus, D. E., Milling Methods and Costs at the Montana Mine Concentrator of the Eagle-Picher Lead Co., Ruby, Ariz.: Bureau of Mines Inf. Circ. 6497, 1931, 14 pp.

[7] Dalton, M. P., Milling Methods and Costs at the Morning Concentrator of the Federal Mining & Smelting Co., Mullan, Idaho: Bureau of Mines Inf. Circ. 6587, 1832, 11 pp.

[8] Zeigler, W. L., Milling Methods and Costs at the Lead Concentrator of the Hecla Mining Co., Gem, Idaho: Bureau of Mines Inf. Circ. 6600, 1932, 16 pp.

[9] Bemis, H. D., Milling Methods and Costs at the Pecos Concentrator of the American Metal Co., Tererro, N. Mex.: Bureau of Mines Inf. Circ. 6605, 1932, 23 pp.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

1. GENERAL DATA—continued

Mine and year	General nature of ore treated	Mill heads			General method of treatment
		Lead, percent	Zinc, percent	Silver, ounces per ton	
San Guillermo (El Potosi), Mexico, 1931 and 1932 (150 days).[10]	Marmatite, argentiferous galena, pyrite, pyrrhotite, with magnetite and silicates replacing limestone.	[11] 10.91 [12] 11.26	[11] 11.29 [12] 11.48	[11] 8.62 [12] 8.59	Flotation. Roughing in lead section producing finished lead concentrate and tailing. Tailing fed to zinc roughers, concentrate from which is cleaned twice to produce final zinc concentrate.
Page, Idaho, 1930[13]	Intimate mixtures of lead and zinc sulfides in a quartzite gangue with pyrite, siderite, and calcite.	9.44	2.46	4.39	Flotation. Roughing in lead section followed by 3 stages of cleaning. Lead section tailing is zinc section feed. Zinc rougher concentrate cleaned twice.
Bunker Hill and Sullivan, Idaho, *A.* West mill, 1929.[14]	Sphalerite and argentiferous galena with pyrite and siderite in quartzite gangue. Some of the galena is in large masses.	10.42	1.55	4.29	Gravity and flotation. (Plus 35-mesh material treated by jigging and tabling producing finished concentrate. Middlings reground and together with tailings are treated by flotation roughing and two stages of cleaning.)
Bunker Hill and Sullivan, Idaho, *B.* South mill, 1929.[14]	Sphalerite and argentiferous galena with pyrite and siderite in quartzite gangue. Some of the galena and sphalerite in fairly large crystals, but most of it is fine.	4.33	7.56	1.22	Flotation. (Roughing and 2-stage cleaning of both lead and zinc roughing concentrates.)
Chief Consolidated, Utah, 1929.[15]	Oxidized siliceous ores containing lead, silver, gold, and small amounts of copper. Lead minerals are cerussite, galena, and anglesite.	11.36	----------	13.00 (0.099 oz. gold)	Bulk flotation of lead and zinc without sulphidizing (using special reagents for oxidized ores added in stages). Formerly, when treating larger tonnage, tailing was stored and treated by chloride volatilization.

2. CRUSHING PLANT

Mine	Primary crusher				Secondary crushers					Capacity of crushing plant, tons per hour
								Size of product		
	No.	Type	Size	Size of product, inches	No.	Type	Size	Percent	Size, inches or mesh	
Mascot, Tenn.	1	Gyratory	No. 8 Chalmers & Williams.	−5½ to disk crusher	1 [16] 1 1	Disk Rolls do.[17]	48 in. 54 x 20 in. 43½ x 16 in.	17.9 4.5 30.5 5.6	+1¾ in. −6-mesh +3-mesh −200-mesh	98.6 (rate in 1929).

White Bird, Okla	1	Jaw	24 in	−3	1	do	42 in			30+.
					2	do.[18]	36 in		−½ in	
Black Hawk, N. Mex	1	do	15 x 24 in	−2½ to −3	1	Cone	2 ft	4.28	+¼ in	20–25.
								5.95	−200-mesh	
Tybo, Nev	1	do	18 x 30 in	−4	1	do	4 ft		−⅜-in. setting	65.
Ruby, Ariz	1	do	18 in	−2	1	Rolls	42 in		do	
					1	do	32 x 24 in[19]		−5 mm	
Morning, Idaho	1	do	9 x 15 in	4-in. setting	1	Cone[20]	4 ft	3.5	−2+1½ in	
								8.6	−7 mm	
Hecla, Idaho	1	Gyratory	28 x 36 in	5-in. setting	1	Rolls in closed circuit with screens.	42 x 16 in	16.2	−30+18 mm	150.
	2	do		1¼-in. setting				1.0	−200-mesh	
Pecos, N. Mex.[21]	1	Jaw	18 x 30 in		1	Rolls	54 x 20 in	6.0	+1 in	
	1	Gyratory	10 in	−1½				7.3	−200-mesh	
San Guillermo, Mexico	1	Jaw[22]	36 x 24 in	3-in. setting	1	Cone[23]	5½ ft	4.44	+½ in	80+.
								3.26	−200-mesh	
Page, Idaho	1	do	9 x 15 in	2.67 percent +3 in	1	do	4 ft	18.82	+1 in	
								4.30	−200-mesh	
					1	Rolls	14 x 36 in	4.7	+½ in	
								17.4	−200-mesh	
Bunker Hill, Idaho *A.* West mill.	2	Gyratory	No. 5 Kennedy	2½-in. setting	1	Disk	36 in			110 normal; 165 capacity.
					1	Rolls	36 x 14 in		7 mm. setting	
Bunker Hill, Idaho *B.* South mill.	2	do	Comet. "D"	do	1	Cone	4 ft			35 normal; 50 capacity.
					1	Rolls	36 x 14 in		7 mm. setting	
Chief Consolidated, Utah	1	do	13 x 45 in	84 percent −2 in	1	do	60 x 24 in	7.41	+1.05 in	65.
								2.26	−200-mesh	

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[10] Mehring, C. A., Bacon, F. C., and Wiser, Oba, Milling Methods and Costs of the San Guillermo Concentrator, of the Compania Industrial "El Potosi," S. A., San Guillermo, Chihuahua, Mexico: Bureau of Mines Inf. Circ. 6706, 1933, 23 pp.

[11] 1931.

[12] 1932.

[13] Price, G. S., Milling Methods and Costs at the Page Concentrator of the Federal Mining & Smelting Co., Kellogg, Idaho: Bureau of Mines Inf. Circ. 6590, 1932, 6 pp.

[14] Handy, J. S., Milling Methods and Costs at the Northern Idaho Mills of the Bunker Hill & Sullivan Mining & Concentrating Co.: Bureau of Mines Inf. Circ. 6314, 1930, 53 pp.

[15] Wigton, G. H., Mining Methods and Costs at the Concentrator of the Chief Consolidated Mining Co., Eureka, Utah: Bureau of Mines Inf. Circ. 6320, 1930, 18 pp.

[16] Tertiary.

[17] Regrind rolls.

[18] In closed circuit with ½-inch trommel screens.

[19] Finishing rolls.

[20] Picking belt precedes cone crusher.

[21] Crushing plant at mine. (See Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Bureau of Mines Inf. Circ. 6368, 1930, 21 pp.)

[22] At mine.

[23] At mill.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

3. GRINDING

Mine	Grinding mills						Circulating load, per cent	New feed, tons per hour	Skeleton size analysis			
					Grinding media				Feed		Product	
	No.	Kind	Size	Circuit	Kind	Size, inches			Per cent	Inch or mesh	Per cent	Mesh, classifier overflow
Mascot, Tenn.	1	Rod mill [24]	7- x 10-ft.	Open, followed by drag classifier.		3- x 115-in.	0	29				
	2	Ball mills [25]	5½- x 22-ft.	Open	Steel	2½	0				1.7 48.8	+35 −200
	1	Pebble mill [25]	7- x 12-ft.	do			0					
White Bird, Okla.	1	Ball mill	6-ft. x 36-in.	Closed with screen and tables.	Forged steel	3		10		−⅛-in.	46.0	−50
Black Hawk, N. Mex.	1	do	6- x 6-ft.	Closed with rake classifier.	Chrome steel	4	300	±7	4.28 46.39	+¼-in. −¼+8-mesh	7.0 75.0	+100 −200
Tybo, Nev.	2	do	8-ft. x 48-in.	do	Steel	3	±400			−⅜	5.09 68.85	+65 −200
Ruby, Ariz.	1	do	8-ft. x 36-in.	Closed with belt drag.	Cast iron	3				−5-mm	30	−200
Morning, Idaho	5	do. [24]	do	Partly closed with drags.	do	3 and 4		±50	3.5 8.6	−2-+1½-in. −7-mm	.2	+65
	2	do. [26]	do	Closed with drags	do	3					81.4	−200
	2	do. [27]	8-ft. x 22-in.	Closed	Steel	1¼						
Hecla, Idaho	3	Rolls [27]	26- x 15-in.	Open			0					
	1	Ball mill [24]	5- x 6-ft.	Closed with drag and screen.								
	1	do. [26]	6-ft. x 22-in.	do							.3 70.7	+48 −200
	3	do	8-ft. x 36-in.	Closed			300					
Pecos, N. Mex.	2	do	7- x 5-ft.	Closed with rakes	Forged steel	4½	400	+25	6.0 7.3	+1-in. −200-mesh	2.2 80.8	+65 −200
San Guillermo, Mexico.	4	Rod mill [24]	7- x 12-ft.	do	Steel	3-in. x 12-ft.		96	4.44 3.26	+½-in. −200-mesh	2.5 48.0	+35 −200
	2	do. [24]	5- x 12-ft.	do	do	do						
Page, Idaho	1	Ball mill [24]	8-ft. x 36-in.	Closed with classifier								
	1	do. [24]	6-ft. x 4½-ft.	Partly closed				12.5	23.0 9.6	+½-in. −200-mesh	.21 72.10	+48 −200
	1	do. [26]	6-ft. x 16-in.	Open			0					
Bunker Hill, Idaho, A. West mill, 1929.	1	do	8-ft. x 36-in.	Low middling, closed	Cast iron	3	334	8.75	.9 .1	+3-mesh −200	72.1	−200
	1	do	6-ft. x 22-in.	High middling, open	do	3	0	7.75	1.5 .2	+3-mesh −200	69.3	−200
	2	do	6 ft. x 22-in.	Closed	do	2½	860	5.4				

Bunker Hill, Idaho, *B.* South Mill, 1929.	2	do.[24]	8-ft. x 36-in.	Partly closed	do.	3	96	17.0	2.9 22.1	+3-mesh −200	.2 91.9	+65 −200
		do.[26]	6-ft. x 22-in.	Closed	do.	3	419	8.35	0.3 15.3	+10-mesh −200	.7 87.8	+100 −200
Chief Consolidated, Utah.	1	do.	8-ft. x 48-in.	Closed with rakes	Chrome steel	5	340	7.0	7.41 2.26	+1.05-in. −200-mesh	1.98 73.27	+65 −200

4. CONCENTRATION—FIRST STAGES

Mine	Method	Feed						Products					
		Source	Tons per hour	Assay, percent		Skeleton size		Lead concentrate			Zinc concentrate		
				Lead	Zinc	Per-cent	Inch or mesh	Assay, percent		Disposition	Assay, percent		Disposition
								Lead	Zinc		Lead	Zinc	
Mascot, Tenn.	Jig roughing[24]	Roll-screen undersize.	75	0.0	3.23	30.5 5.6	+3 mesh −200	None		None			Hutch to cleaning jigs.
	Bull jigging[26]	Roughing jig tailing oversize.				+⅛ x 1 in.		do.		do.			Hutch to regrind rolls.
	Sand jigging	Fines from regrind rolls.					−3 mesh	do.		do.			Hutch to cleaning jigs.
	Flotation roughing[16].	Thickened ball-mill product and drag overflow.	36	.0	3.46	1.7 48.8	+35 mesh −200	do.		do.			To cleaning cells.
White Bird, Okla.	Jig roughing[24]	Deslimed roll-screen product.	31			14.9 2.1	+3 mesh −100	do.	None.	To cleaning jig.[28]			To cleaning jig.[28]
	Sand jigging[25]	Reground rougher middling and cleaner-jig tails.					−⅛ in.	do.	None.	First hutch to head of cleaner.[28]			First hutch to head of cleaner;[28] secondi and third hutch finshed concentrate.
	Tabling[15]	Sand-jig tailing and slime from sand-jig feed via drag-belt classifier.			10.50		−2 mm	do.	None.	None			Final.

[16] Treasury.
[24] Primary.
[25] Secondary, only one of 3 mills used in regular operation.
[26] Secondary.
[27] Middling regrind.
[28] Lead and zinc bulk concentrate.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

4. CONCENTRATION—FIRST STAGES—continued

Mine	Method	Feed						Products					
		Source	Tons per hour	Assay, percent		Skeleton size		Lead concentrate			Zinc concentrate		
				Lead	Zinc	Percent	Inch or mesh	Assay, percent		Disposition	Assay, percent		Disposition
								Lead	Zinc		Lead	Zinc	
Black Hawk, N. Mex.	Flotation roughing, 2 sections.	Classifier overflow (via conditioner).	7±	2.40	11.9	7.0 75.0	+100 mesh −200			First section to lead cleaners.			
	Flotation roughing, 2 sections, zinc section.	Lead rougher tailing from zinc conditioners.								To zinc cleaners.			
Tybo, Nev.	Flotation roughing, 16-cell machine.	Classifier overflow	13.3	5.95	4.14	5.09 68.85	+65 mesh −200			From first 3 to 5 cells, to lead-cleaning cells.			
	Flotation roughing, zinc section, 16-cell machine.	Tailing from lead circuit.		1.85	4.17	6.12 66.54	+65 mesh −200 mesh						First 4 cells to zinc cleaning cells.
Ruby, Ariz.	Flotation roughing, [24].	Classifier and thickner overflows.	10.75	5.13	5.98			57.05	10.06	Final.			
	do.[26]	Middling from primary.								Returned to head of circuit.			
	Roughing zinc section.[24]	Tailing from secondary of lead section.									2.79	52.36	Final.
	do.[26]	Middling from primary.											Returned to zinc conditioner.
Morning, Idaho.	Flotation roughing lead section (6-cell machines).	Ball-mill classifier overflows.	50	9.2	6.7	.2 81.4	+65 mesh −200			First 3 cells to cleaners.			
	Flotation roughing zinc section (8-cell machines)[24].	Lead section tailing.											First 4 cells to zinc cleaners.

	Flotation roughing zinc section.[26]	Tailing from primary.											To zinc regrind mill.
	Flotation roughing and cleaning.[16]	Regrind mill								Cleaned rougher[16] concentrate to lead section.			
Hecla, Idaho	Hand sorting	Screened crusher product.						50.0±		Shipping product.			
	Jigging	Sized [29] feed from 42- x 16-inch rolls.	22±	8.24	.86	16.2 1.0	+30−18 mm −200 mesh	52.02 (27.1 oz silver).	3.50	All but last jig compartments are final.			
Pecos, N. Mex.	Flotation (mica section).	Classifier overflow	25±	4.93	15.41	2.2 80.8	+65 mesh −200			Mica concentrate refloated and tabled.			
	Tables (mica section).	Cleaned mica flotation concentrate.								To lead section.			
San Guillermo, Mexico, 1932.	Flotation lead section (14-cell machines).	Rod mill classifier overflows.	96	11.26	11.48	2.5 48.0	+35 −200	67.94 (48.09 oz. silver).	3.71	Final			
	Flotation roughing, zinc section.	Lead section tailing										40.0±	To first zinc cleaner.
Page, Idaho, 1930.	Flotation roughing, lead section (8-cell machines).	Classifier overflow	12.5	9.44	2.46	.21 72.10	+48 mesh −200			First 3 cells to first cleaner.			
	Flotation roughing, zinc secion.	Lead rougher tailing.											First 3 cells to first zinc cleaner.
Bunker Hill, Idaho, *A*. West Mill.	Jigging and tabling in series.	Rolls-classifier underflow.	9.75			[30] 2.9 .9	+3 mesh −48 mesh			First 2 hutches final, others to fine grinding.			
	Tables		.70 per table.			[31] 2.2 4.0	+14 mesh −200 mesh			Final to bin			
Bunker Hill, Idaho, *B*. South Mill.	Selective flotation, roughing, lead section.	Classifier overflow	17.3	4.33	7.56	.7 87.8	+100 −200			First 4 cells to first lead cleaner.			
	Flotation roughing, zinc section.	Lead section tailing											First 4 cells to first zinc cleaner.
Chief Consolidated, Utah, 1929.	Flotation 32 cells [32] *A*. roughing.		7.0	11.36 (13.0 oz. silver).	1.65	1.98 73.27	+65 −200			Froth from cells 10–16 to cleaner cells.			

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[16] Tertiary.
[24] Primary.
[26] Secondary.
[29] Jigging of 5 separate sizes, viz., −30+48 mm.; −18+12 mm.; −12+7 mm.; −7+3 mm.; −3 mm.+16 mesh.
[30] Jig feed.
[31] Table feed.
[32] First 9 cells are cleaners; next 7 make rougher concentrate, last 16 make scavenger concentrate.

Table 60.—*Data on lead and zinc ore concentrators*—Continued

4a.—CONCENTRATION, FIRST STAGES

Mine	Method	Products—Continued						
		Middling			Tailing			
		Assay, percent		Disposition	Assay, percent		Disposition	
		Lead	Zinc		Lead	Zinc		
Mascot, Tenn.	Jig roughing	0.0		To regrind rolls [33]	0.0		Oversize to bull jigs.	
	Bull jigging	.0					Final tailing.	
	Sand jigging	.0					To rod mill.	
	Flotation roughing	.0		To middling re-treatment flotation				
White Bird, Okla.	Jig roughing			To 36-in. regrind rolls; to sand jig		1.24	Deslimed and rejected (75 percent of total mill feed).	
	Sand jigging			4th and 5th hutch to cleaning jig			To tables.	
	Tabling			Lead-zinc to separate recleaner. Zinc to head of tables.			First 3 tables plus table-screen oversize dewatered and reground; thence to head of tables.	
						1.32	Last 4 tables; final tailing to waste.	
Black Hawk, N. Mex.	Flotation roughing			Second section of rougher back to conditioner.			To zinc conditioners (two in series).	
	Flotation roughing, 2 sections						Final tailing to pond.	
Tybo, Nev.	Flotation roughing			Cells 6 to 10 returned to head of rougher; cells 11 to 13 to cell 8; from cells 14–16 to cell 11.	1.85	4.17	To zinc circuit.	
	Flotation roughing, zinc section			Cells 5–8 to head of zinc rougher; cells 9–16 to cell 5.	1.85	1.01	Final tailing to pond.	
Ruby, Ariz.	Flotation roughing [24]			To secondary flotation				
	do.[26]						To zinc section conditioner.	
	Roughing, zinc section [24]			To secondary flotation (zinc section)	.19	1.26	Final to tailing dam.	
	do.[26]							
Morning, Idaho	Flotation roughing, lead section			From last 3 cells to lead circuit regrind mill.			To zinc section.	
	Flotation roughing, zinc section [24]			Concentrate from last 4 cells to head of zinc section.			To secondary zinc roughers.	
	do.[26]						Final, to waste.	
	do.[16]						To head of zinc roughers.	
Hecla, Idaho	Sorting			Oversize to 2 gyratories; undersize to ore bin.				
	Jigging	6.30	3.00	From last compartment of each coarse jig to middling rolls and back to head of circuit	.60	.90	To dewatering, thickening, 3 ball mills and flotation rougher (jig tailing section).	
				From fine jigs to ball mills and selective flotation section.				

Pecos, N. Mex.	Flotation (mica section)			Returned to head of section			To lead section.
	Tabling (mica section)						Mica tailing to waste.
San Guillermo, Mexico.	Flotation (lead section)						To head of zinc roughers.
	Flotation roughing (zinc section)				.50	2.19	To waste.
Page, Idaho, 1930	Flotation roughing (lead section)			Cells 4 to 8 returned to grinding section			To head of zinc section.
	Flotation roughing (zinc section)			Cells 4 to 8 returned to head of zinc circuit.	[34] .95	.44	Final to waste.
Bunker Hill, Idaho, *A.* West mill.	Jigging			From classifier overflows to simple flotation.			To selective flotation.
	Tabling	3.0	4.9	To fine grinding			To fine grinding.
Bunker Hill, Idaho, *B.* South mill.	Selective flotation roughing-lead section.			Last 5 cells return to head of rougher			To zinc rougher.
	Flotation roughing-zinc section			Last 5 cells return to head of zinc rougher	[34] .52	[34] 1.19	To waste.
Chief Consolidated, Utah, 1929.	Flotation roughing			Cells 17–32 froth back to head of circuit	[35] 1.30 [35] 4.30 oz. silver.	[35] 1.44	To waste after thickening.

5. CONCENTRATION, FINAL STAGES

Mine	Method	Feed						Products					
		Source	Tons per hour	Assay, percent		Skeleton size		Lead concentrate			Zinc concentrate		
				Lead	Zinc	Percent	Inches or mesh	Assay, percent		Disposition	Assay, percent		Disposition
								Lead	Zinc		Lead	Zinc	
Mascot, Tenn.	Jig cleaning	Hutch from rougher and sand jigs.		0.0				None			0.00	59.08	First 4 cells final.
	Flotation middling re-treatment.	Rougher middling						do			See 5*a*.	See 5*a*.	See 5*a*.
	Flotation cleaning	Rougher concentrate						do			0.00	61.50	Final.
White Bird, Okla.	Jig cleaning	Rougher jig concentrate and first hutch of sand jig.				2.4 1.0	4 mesh 200 mesh	74.00 first hutch.	5.20	Final	0.07 last 4 hutches.	59.80 last 4 hutches.	Final.
	Table cleaning	Lead-zinc table middling.						74.00	5.20	Final lead concentrate.	None	None	
	Flotation roughing, 3 machines in series.	Thickened slimes from gravity plant dewatering cones and drags.			9.40			None	None	None	0.36 first machine and part of second.	61.50 first machine and part of second.	Final to filter.

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[16] Tertiary.
[24] Secondary, only 1 of 3 mills used in operation.
[16] Secondary.
[33] In closed circuit with drag-belt classifier and Hummer screens.
[34] Calculated.
[35] By difference.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

5 CONCENTRATION, FINAL STAGES—continued

Mine	Method	Feed						Products					
		Source	Tons per hour	Assay, percent		Skeleton size		Lead concentrate			Zinc concentrate		
								Assay, percent		Disposition	Assay, percent		Disposition
				Lead	Zinc	Percent	Inches or mesh	Lead	Zinc		Lead	Zinc	
Black Hawk, N. Mex.	Flotation cleaning, lead circuit, 2 stages.	Lead concentrate from first roughing section.						58.37	11.57	Final			
	Flotation cleaning, zinc circuit, 2 stages.										1.81	54.77	Final.
Tybo, Nev.	Flotation cleaning, lead section.	First 3 to 5 cells of rougher.						59.90	4.00	Final to thickener.			
	Flotation cleaning, zinc section.	First 4 cells of rougher.									2.55	47.60	Final to thickener.
Ruby, Ariz. (See concentration, first stages (4).)													
Morning, Idaho	Flotation cleaning, lead section, 3 stages.	Primary rougher concentrate, first 3 cells.						74.2	5.8	Final to thickener.			
	Flotation cleaning, zinc section, 3 stages.	Primary rougher concentrate, first 3 cells.									2.8	56.2	Final to thickeners.
Hecla, Idaho	(1) Flotation	Primary slime								Final			
	(2) Flotation (selective):												
	(*a*) Lead section	Classifier overflow	6.8±	9.70	2.30	.3 70.7	+48 mesh −200 mesh	60.9	6.6	Final			
	(*b*) Zinc section	Lead section tailing									5.3	52.0	Final.
	(3) Flotation:												
	(*a*) Lead section	Jig tailing, 3 ball mills.	17.4±	.60	.90					Final to thickeners.			
	(*b*) Zinc section	Lead section tailing											Final to thickeners.

Pecos, N. Mex.	Flotation, lead section (16-cell machines).	Mica section tailing and table concentrate enters 5th cell.						37.81	13.47	First 4 cells final.			
	Flotation roughing, zinc section.	Tailing from lead section.											First 5 cells to cleaner.
	Flotation, zinc cleaning.	First 5 rougher cells.									1.63	54.45	Final after tabling.[36]
San Guillermo, Mexico, 1932.	Flotation cleaning (zinc).[37]	Rougher concentrate			40.0±							45.0±	To second cleaner.
	do.[38]	First cleaner concentrate.			45.0±						0.93	46.53 (1.72 oz. silver).	Final.
Page, Idaho, 1930.	Flotation cleaning, lead section, 3 stages.	Rougher froth, first 3 cells.						68.1 (30.7 oz. silver).	7.9	Final from third stage.			
	Flotation cleaning, zinc section, 2 stages.	Rougher froth from first 3 zinc cells.									3.8	51.6 (5.6 oz. silver).	Final from 2d stage.
Bunker Hill, Idaho, A. West mill.	Selective flotation roughing, lead section.	Gravity tailing, reground middling.	8.33 capacity.			0.3 65.6	+48-mesh −200-mesh			First 4 cells to first cleaner.			
	Simple flotation	Primary slimes	8.33 capacity.			0.5 95.6	+100 mesh −200 mesh						
	Flotation cleaning, lead section, 2 stages.	First 4 rougher cells	1.4							Second stage final.			
	Flotation roughing, zinc section.	Primary slimes and lead section tailing.	[39] 20		1.49								First 4 cells to cleaner.
	Flotation cleaning, zinc section.	First 4 cells of rougher.	1.25										First 4 cells final.
Bunker Hill, Idaho, B. South mill.	Flotation cleaning, lead section, 2 stages.	First 4 rougher cells.	5.2 capacity.					70.33	8.19	Second stage final.			
	Flotation cleaning, zinc section, 2 stages.	First 4 zinc rougher cells.	5.2 capacity.								3.58	51.0	2d stage final.
Chief Consolidated, Utah, 1929.	Flotation cleaning, first 9 cells.	Cells 10–16						33.43 (32.05 oz. silver).	2.12 (0.225 oz. gold).	Final to thickener.			

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[36] Zinc concentrate tabled to remove a high-grade gold-copper product, which is added to the lead concentrate.

[37] First stage.

[38] Second stage.

[39] Simple flotation plus selective flotation sections.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

5*a*. CONCENTRATION, FINAL STAGES

Mine	Method	Products, Continued					
		Middling			Tailing		
		Assay, percent		Disposition	Assay, percent		Disposition
		Lead	Zinc		Lead	Zinc	
Mascot, Tenn.	Jig cleaning			5th hutch returned to 1st cell			To rod mill.
	Flotation middling re-treatment			Return to mixer ahead of rougher	0.0	0.09	Final tailing.
	Flotation cleaning			do			
White Bird, Okla.	Jig cleaning			2d hutch to head of cleaner jig			Joins rougher middling, to regrind rolls to sand jigs.
	Table cleaning			To ball-mill regrind			
	Flotation roughing			Froth from part of 2d and all of 3d rougher returned to head of 1st machine.		.60	To waste.
Black Hawk, N. Mex.	Lead flotation cleaning			Returned to conditioner ahead of roughers.			
	Zinc flotation cleaning			Returned to conditioners ahead of zinc section.			
Tybo, Nev.	Flotation cleaning, lead section			To head of rougher			
	Flotation cleaning, zinc section			do			
Ruby, Ariz. (See Concentration, first stages (4*a*))							
Morning, Idaho	Flotation cleaning, lead section			To lead regrind mill, to classifiers of primary grinding section.			
	Flotation cleaning, zinc section			To head of zinc roughers			
Hecla, Idaho	1. Flotation (primary slime)						To waste.
	2. Flotation (selective):						
	(*a*) Lead section						To zinc section.
	(*b*) Zinc section				.54	.13	To waste.
	3. Flotation (jig tailing):						
	(*a*) Lead section						To zinc section.
	(*b*) Zinc section						Final, to waste.
Pecos, N. Mex.	Flotation (lead section)			Cells 5 to 13 to first cleaning cell; cells 14 to 16 to cell 13.			To zinc section.
	Flotation roughing, zinc section			Last 11 cells return to zinc conditioner	.76	1.33	To zinc scavenger (used principally for pilot purposes) then to waste.
	Flotation, zinc cleaning						Thickened and returned to zinc conditioner.

San Guillermo, Mexico, 1932.	Flotation cleaning, first stage (zinc)			Returned to zinc roughers			
	Flotation cleaning, second stage			do			
Page, Idaho	Flotation cleaning, lead section			1st stage to head of roughers; 2d stage to head of 1st stage; 3d stage to head of 2nd stage.			
	Flotation cleaning, zinc section			1st stage to head of zinc section; 2d stage to head of 1st stage.			
Bunker Hill, Idaho, *A*. West mill.	Selective flotation roughing, lead section			Last 5 cells return to head of rougher			To zinc rougher.
	Simple flotation (slime section)			do	.61	.49	Final tailing.
	Flotation cleaning, lead section						1st stage to head of rougher, 2d stage to head of 1st stage.
	Flotation roughing, zinc sections			Last 4 cells, return to head of zinc section	.35	.35	To waste.
	Flotation cleaning, zinc sections						1st stage to head of zinc rougher, 2d stage to head of 1st stage.
Bunker Hill, Idaho, *B*. South mill.	Flotation cleaning lead section						1st stage return to head of lead rougher, 2d stage to head of 1st stage.
	Flotation cleaning, zinc section						1st stage to head of zinc rougher, 2d stage to head of 1st stage.
Chief Consolidated, Utah, 1929.	Flotation cleaning, first 9 cells						To cell no. 10.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

6. SUMMARY OF CONCENTRATION

Mine	Ratio of concentration				Grade of concentrates, percent				Total recovery, percent		
	Gravity		Flotation		Lead concentrate		Zinc concentrate		Lead [40]	Zinc [40]	Silver
	Lead	Zinc	Lead	Zinc	Lead	Zinc	Zinc	Lead			
Mascot, Tenn., 1929	None	60 to 1	None		None	None	60.19	0.00		90.15	
White Bird, Okla	509 to 1 [41]	16.5 to 1 [41]	do	36.9 to 1 [41]	74.00	1.07	60.13	.16	85.60	84.3	
Black Hawk, N. Mex	None	None	36 to 1 [41]	63 to 1 [41]	58.37 (25.22 oz. silver).	11.57	54.77	1.81	81.86	90.69	62.95 (copper 23.45).
Tybo, Nev	do	do	14.1 to 1	15.9 to 1	59.90 (87.1 oz. silver).	4.00	47.60 (14.28 oz. silver).	2.55	74.0	79.2	74.9
Ruby, Ariz	do	do	12 to 1	12.8 to 1	57.05 (0.674 oz. gold and 74.02 oz. silver).	10.06	52.36 (0.07 oz. gold, 12.4 oz. silver).	2.79	[42] 92.59 [43] 4.23	[42] 14.02 [43] 68.29	[42] 75.49 [43] 11.92
Morning, Idaho	do	do	9.1 to 1	10.0 to 1	74.2 (24.9 oz. silver).	5.8	56.2 (4.8 oz. silver).	2.8	[42] 91.0 [43] 3.1	[42] 9.7 [43] 85.7	[42] 81.1 [43] 14.2
Hecla, Idaho	*Sorting;* 17.8 to 1 for shipping ore 6.24 to 1 for waste rock				Not given.						
	Jigging: 7.4 to 1 [44]		*Selective section* 6.8 to 1 [45] *Jig tailing section:* 58 to 1 [46]	38 to 1	53.73 (29.01 oz. silver).	4.5	46.70 (10.1 oz. silver).	9.30	[42] 97.4 [43] .7	[42] 65.0 [43] 27.7	[42] 96.1 [43] 1.4
Pecos, N. Mex			9.35 to 1 [41]	4.17 to 1 [41]	37.81 (19.98 oz. silver; 0.768 oz. gold).	13.47	54.45 (3.05 oz. silver; 0.036 oz. gold).	1.63	[42] 81.97 [43] 7.91	[42] 9.35 [43] 84.64	[42] 63.43 [43] 21.73
San Guillermo, Mexico, 1932.			6.4 to 1 [41]	4.9 to 1 [41]	67.94	3.71	46.53	.93	[42] 94.76 [43] 1.69	[42] 5.08 [43] 85.60	[42] 87.62 [43] 4.10
Page, Idaho, 1930			7.96 to 1 [41]	47.0 to 1 [41]	68.1 (30.7 oz. silver).	7.9	51.6 (5.6 oz. silver).	3.8	[42] 90.6 [43] .8	[42] 40.2 [43] 44.6	[42] 87.8 [43] 2.7
Bunker Hill, Idaho, *A.* West mill, 1929.			6.27 to 1 [47]	131 to 1 [47]	58.19 (24.11 oz. silver).	3.13	47.37	3.77 (5.9 oz. silver).	[42] 89.12 [43] .28	[42] 32.15 [43] 23.29	[42] 89.70 [43] 1.05
Bunker Hill, Idaho, *B.* South mill, May–Dec., 1929.			20.3 to 1	8.3 to 1	70.33 (13.58 oz. silver).	8.19	51.00	3.58 (3.39 oz. silver).	[42] 80.04 [43] 10.00	[42] 5.33 [43] 81.60	[42] 54.77 [43] 33.57
Chief Consolidated, Utah			3.19 to 1		33.43 (32.05 oz. silver).	2.12 (0.225 oz. gold).			[42] 92.14	[42] 40.19	77.26 (gold, 71.43).

7. DEWATERING, FILTERING, TAILING DISPOSAL

Mine	Concentrates						Tailings
			Dewatering		Filtering		
	Source	Moisture, percent	Method	Moisture in product, percent	Method	Moisture in cake, percent	Method of disposal
Mascot, Tenn.	Jigs		Storage bins to Lowden drier				Thickened and laundered to limestone plant.
	Flotation		Thickening	37.0	Drum filter Lowden drier	7.5 .1	Pumped to limestone plant.
White Bird, Okla.	Jigs and tables		Drained in boxes and shoveled into cars.	4.0			Tailing elevators (2 in series).
	Flotation				Disk filter	10.5	
Black Hawk, N. Mex.	Lead section	95+	Thickening	50±	Leaf filter	10.0	Laundered to pond.
	Zinc section	95+	do	50±	do	11.0	
Tybo, Nev.	Lead section	90±	do	30–40	Drum filter	?	Do.
	Zinc section	90±	do	30–40	do	?	
Ruby, Ariz.	Lead section		None		Disk filter	7.67	Pumped 800 feet to either of two ponds. Water reclaimed for use in mill.
	Zinc section		do		do	9.58	
Morning, Idaho	Lead section		Thickening	20	Drum filter		
	Zinc section		Thickening; lime added to aid settling.	25	do		
Hecla, Idaho	Lead (jig)		Drag dewaterer to bins		None		
	Lead (flotation)		Thickening	40	Drum filter	7.5	
	Zinc		do		do		
Pecos, N. Mex.	Lead	85 to 90	do	65–75	do	7.0 to 8.0	Laundered 1 mile to dam.
	Zinc	72	do	65–75	do	7.5 to 9.0	
San Guillermo, Mexico	Lead		Drag classifier to thickening		Disk filters	7.5	Laundered 3,174 feet to sand pumps to dam.
	Zinc		do		Drum filter	7.0	Water pumped back to mill.
Page, Idaho	Lead		Thickening		do		
	Zinc		do		do		
Bunker Hill, Idaho, *A*. West Mill, 1929	Lead (gravity)		Drag		Jigs, none; tables, drum filter		Dewatered in large drag; thickened and water reclaimed.
	Lead (flotation)				Drum filter		
	Zinc		Thickening		do		
Bunker Hill, Idaho, *B*. South Mill.	Lead		None		do		
	Zinc		do		do		
Chief Consolidated, Utah.	Flotation cleaner froth	75	Thickening	50	Disk filters	Summer, 15.0; winter, 16.0	Thickened; overflow water returned to mill, thickened pulp piped to waste.

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[40] Figures in these columns include recovery of zinc in lead concentrate and of lead in zinc concentrate as reported in original sources of information and to balance the data. It should be noted, however, that zinc in lead concentrates is not paid for by the smelters and when present in excess of prescribed maximum amounts takes a penalty charge, and that zinc smelters ordinarily pay for none or only part of the lead content of zinc concentrates and then at a reduced price.

[41] Total mill feed divided by final concentrate produced.

[42] In lead concentrate.

[43] In zinc concentrate.

[44] Based on feed to jigs.

[45] Based on feed to selective section.

[46] Based on feed to jig-tailing section.

[47] Gravity and flotation combined.

TABLE 60.—*Data on lead and zinc ore concentrators*—Continued

8. OPERATING COSTS PER TON OF ORE MILLED

Mine and year	Ore treated per day, tons	Operating costs per ton													
		Crushing	Grinding	Concentrating	Dewatering and filtering	Disposal of tailing	Other	Total	Labor	Power and water	Supplies			Other supplies and general	Total
											Crushing	Grinding	Reagents		
Mascot, Tenn., 1929	1.800	[48] $0.079	$0.053	[49] $0.063 [50] .082	[51] $0.029	$0.005	[52] $0.030	[53] $0.341	--------	--------	--------	--------	--------	--------	$0.341
White Bird, Okla., Aug.-Oct., 1929	312	.086	.078	[49] .123 [50] .090	[54] .065	--------	.062	.504	$0.155	$0.153	$0.049	$0.054	$0.039	$0.054	.504
Black Hawk, N. Mex., Sept.-Oct., 1929	165	.111	.355	.526	.082	[55] .047	.430	1.551	.510	.345	--------	--------	.302	.394	1.551
Tybo, Nev., Mar. 1930	320	[56] .097	.445	.502	.021	--------	.217	1.282	.244	.318	.019	.176	[57] .359	.166	1.282
Ruby, Ariz., Oct. 1, 1929, to Apr. 1, 1930	258	.321	.252	.598	.050	.062	[58] .919	2.202	.587	.360	.095	.103	[57] .363	.694	2.202
Morning, Idaho, 1930	±1,200	[59] .080	[59] .150	[59] .242	[59] .030	--------	.257 [60] .270	1.029	--------	--------	--------	.101	--------	--------	--------
Hecla, Idaho, 1930	900	([61])	--------	--------	--------	--------	--------	--------	[62] .242	[62] .143	([62])	[62] .079	[57] .024	[62] .062	[62] .550
Pecos, N. Mex., 1930	±500	[63] .059	.271	.558	.041	.001	.319	1.249	[62] .330	.231	.059	.155	.295	.179	1.249
San Guillermo, Mexico, 1931	±1,990	[64] .014	[59] .074	[59] .126	[59] .022	[59] .008	.329	.573	--------	.449	--------	--------	--------	--------	--------
San Guillermo, Mexico, 1932 (150 days)	2,255	[64] .017	[65] .157	[65] .189	[65] .029	[65] .011	[65] .101	.504	.112	.247	[64] .005	.046	.070	.024	.504
Page, Idaho, 1930	300	.054	.411	.292	.031	--------	.117	.905	.261	.284	--------	.210	.120	.030	.905
Bunker Hill, Idaho. *A.* West mill, 1929	1,275	.152	.253	[49] .268 [50] .324	--------	--------	.081	1.078	.204	.162	.034	.109	.135	.434	1.078
Bunker Hill, Idaho. *B.* South mill, May-Dec., 1929	416	.261	.354	.970	--------	--------	--------	1.585	.251	.254	.047	.097	[57] .416	.520	1.585
Chief Consolidated, Utah, 1929	168	.123	.462	1.657	.168	.062	.745	3.217	.730	.771	.018	.183	1.211	.304	3.217

[48] Includes tramway operation, $0.0119.

[49] Gravity concentration.

[50] Flotation, including slime pumping.

[51] Includes drying.

[52] Water supply.

[53] Loading commercial limestone tailing, $0.0525 additional.

[54] Elevating.

[55] Sorting.

[56] Includes sorting, $0.018.

[57] Includes other flotation supplies.

[58] Includes pumping water and shop and mill share of overhead expenses: Labor, $0.1675; supplies and renewals, $0.6751.

[59] Except power.

[60] Total power for mill.

[61] Not given.

[62] Does not include crushing plant.

[63] Cost in 1929 (see Inf. Circ. 6368, p. 18).

[64] Crushing at mill cone crusher only; jaw crushing at mine not included.

[65] Includes power.

TOTAL COST OF PRODUCING ORE AND CONCENTRATES

From the viewpoint of mineral economics, the ultimate and significant production cost is that per unit of metal produced rather than the cost per ton of ore treated. This viewpoint is natural to processors of ores whose final product is refined or unrefined metal, whether they be companies engaged solely in smelting, or integrated companies engaged in mining, milling, and smelting operations.

Mine operators, on the other hand, are accustomed to using costs per ton of ore as a basis for measuring the efficiency of their operations and for controlling them. Many companies are engaged only in mining or in mining and milling, and their final product is then either ore or concentrates; hence, cost per ton is the natural and convenient basis to employ. However, the value of ore or concentrate depends on the salable metal it contains, and the measure of profit or loss is the cost per unit of metal in relation to the price at which it can be sold.

For gold and silver mines whose only product is bullion produced from amalgamation or cyanidation, it is relatively simple to set up unit costs on the basis of an ounce of gold or silver, and some gold-mining companies now so state their costs. The operating staff is thus reminded that maximum efficiency results from minimum cost per ounce of metal rather than minimum cost per ton of ore, which should have a salutary effect when such questions as control of dilution and grade of ore in mining and loss of metal in milling are considered.

Cost per ounce of gold [94] is given for a few gold mines in North America in table 17 of Bulletin 363.

Croston [95] recently published much more comprehensive data on costs per ounce of gold, in which are included data from most of the largest gold mines in the world.

At the other extreme from gold mines producing bullion are those mining and treating complex ores containing, for example, silver, gold, lead, zinc, and copper. It is obvious that it would be impracticable, if not impossible, for mines of this type to maintain a mine and mill cost-accounting system in which costs would be prorated accurately to the different metals, and it is doubtful whether such accounts would serve any useful operating purpose. For general analysis of operating over periods of a year or more, proration of costs may be useful. Gardner [96] has presented prorated costs for a number of copper-mining companies in tables 63 and 64 of Bulletin 405.

In the same bulletin, tables 53 to 62, inclusive, present additional data on cost of producing a pound of copper. Table 57 of this series is reproduced below.

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[94] Jackson, Chas. F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bureau of Mines Bull. 363, 1932, 151 pp.

[95] Croston, John J., World Gold Production Costs. Part 1, The Americas: Min. and Met., July 1937, pp. 324–327.

[96] Gardner, E. D., Johnson, C. H., and Butler, B. S., Copper Mining in North America: Bureau of Mines Bull. 405, 1938, 300 pp.

Weighted average costs of American copper production, 1929 and 1930, in cents per pound [1]

	1929 [2]	*1930* [3]
Operating	9.92	9.51
Taxes and interest	1.21	.84
Depreciation	1.07	1.05
Miscellaneous expense	.13	.01
Sundry metals	[4] 1.52	[4] 1.20
Miscellaneous income	[4] 1.17	[4] .48
Net costs after depreciation	9.64	9.73
Copper output (millions of pounds)	2,091	1,370

[1] Strauss, S. D., Copper Production and Costs in 1929 and 1930: Met. and Min. Markets, June 25, 1931, p. 11.

[2] Includes Phelps Dodge, Kennecott, and Cerro de Pasco, which are not in 1930 data.

[3] Includes Consolidated Coppermines, Greene Cananea, and Walker Mining, which are not in 1929 data.

[4] Credit.

Many companies mining ores of the base metals containing precious metals that are recovered as a byproduct, credit base-metal production costs with the return from the precious metals. While this is a convenient practice for general accounting purposes, it is of little use for cost control at the mine.

Costs per pound of lead and zinc (not including smelting) are given for seven companies in table 32 of Bulletin 381.[97]

The following brief discussion and tables giving production and overhead costs per ton are presented, therefore, with due recognition of the fact that the ultimate economy of a mining enterprise depends upon the cost per unit of metal and the net value of the metals produced.

Cost data given in the preceding sections of this bulletin covered operating costs only, that is, expense at the mines and mills—wages and salaries, including workmen's compensation; costs of mine and mill supplies and power; maintenance; and mine and mill office salaries and expense. The circulars from which most of these data were abstracted were intended to cover only operating costs under direct control of the resident management with a view to showing the relationship between (1) natural and economic conditions influencing the methods used and the results obtained, (2) the methods and practices employed under the conditions described, and (3) the resulting operating costs.

Other costs must be added, however, to determine the ultimate cost of production. These include principally salaries of corporation officials and other head-office expense, depreciation, interest charges, legal expense, and taxes. Since these were included in only a few of the information circulars used as a basis for this bulletin, published annual reports of a number of mining companies have been resorted to for the data in table 61, which follows.

In this table, data from annual reports of a few companies have been selected to illustrate broadly the variations in the ratio of overhead to operating costs; many companies do not publish cost data in enough detail to allow distribution under the headings used in the table. Because of variations in accounting methods of different companies, the distribution of some cost items under these headings is somewhat arbitrary in certain instances, most of which are covered by footnotes.

Operating costs in the table cover all exploratory work (except exploration of outside properties), development, mining, crushing and milling, workmen's compensation, and all expense incurred at the mine and mill, including local administrative costs such as salaries of mine executives and office staff. In some of the reports, workmen's compensation is carried as an overhead cost at the executive offices of the company, but since it is in reality an additional labor cost, it has been included in the table (where the reports permitted separation) as an operating cost.

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[97] Jackson, Chas. F., Knaebel, John B., and Wright, C. A., Lead and Zinc Mining and Milling in the United States; Current Practices and Costs: Bureau of Mines Bull. 381, 1935, 204 pp.

Most of the reports include property and local taxes and insurance with operating costs, but in the table they have been included with income and other taxes.

Depreciation is included as an overhead cost directly chargeable to production, since it is contemplated that the sums set aside are to be used for replacement of worn-out or obsolete equipment to permit continuing efficient production.

Amortization items have been added to the table but are not considered a part of the total production cost, although it is recognized that funds for amortization of the investment must come from the profits realized from production and sale of ore or metal. Amortization will be discussed briefly in a later section of this bulletin and is introduced at this point merely to call attention to the fact that it is an item that must be considered with costs and realization from sales in order to determine profit or loss on the operation.

Exploration of outside properties is considered here to be a use of surplus to prolong the life of the enterprise by discovering additional reserves of ore rather than a production-cost item. Thus, instead of taking funds from surplus for investment in bonds or other securities, which, together with accumulated interest and dividends, provide a sinking fund for ultimate return of the invested capital, many mining companies employ part of their surplus in an endeavor to develop new mines. Amortization or sinking funds are provided by allowances for depletion and may be accumulated by charging off a fixed amount per ton of ore or unit of metal produced or by annual charges. Only a few of the reports from which data are taken for the table showed charges for depletion, and some merely set up a charge for amortization of plant and buildings. Some of the older companies probably have charged off all development, including major shaft-sinking and other projects, in the years during which the expenditures were incurred, whereas others have not, so that the data in the depletion column of the table are not complete or comparable as between different companies.

Interest received and other income often are reported as a credit against production costs, but because of the differences in methods of accounting and in the sources from which such income is derived by different companies, and because in most instances it is not a direct result of the year's ore-production operations, the figures are given under "amortization items" in order to make the net of all items in the table balance the figures in the annual reports.

The foregoing suggests the subject of accounting systems for mining companies, which is not within the scope of the present bulletin. In this connection, reference is made to an earlier Bureau of Mines publication.[98]

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[98] Keller, Albert E., Accounting System and Office-Management Procedure for Medium-Size Metal Mines: Bureau of Mines Bull. 372, 1933, 84 pp.

TABLE 61.—*Total costs of producing ore, concentrates, or bullion* [1]

Company and year	Tons mined or milled	Operating costs per ton mined or milled						Overhead costs per ton						Income tax	Amortization items		
		Mining [2]	Milling	General			Total	Depreciation	Interest paid	Taxes (except income taxes)	Other	Total			Exploration [3]	Depletion	Interest received and other income credit
				Surface	Compensation	Other						Amount	Percent of operating cost				
Hollinger, Ontario:																	
1930	[4] 1,625,868	$2.983	$0.636	$0.042	$0.085	$0.244	$3.990	$0.002	--------	$0.082	$0.027	$0.111	2.80	$0.096	$0.071	--------	$0.384
1933	[4] 1,727,102	3.034	.622	.046	[5] .262	.198	4.162	.113	--------	.146	.032	.291	7.00	.259	.036	--------	.093
1935	[4] 1,837,153	3.255	.647	.057	[6] .148	.243	4.350	.012	--------	.096	.128	.236	5.42	.370	.123	--------	.120
1937	[4] 1,719,199	3.474	.658	.054	[7] .197	.252	4.635	.096	--------	.115	.202	.413	8.91	.467	.237	--------	.116
Teck-Hughes, Ontario, fiscal year ended:																	
August 31, 1930	[4] 340,675	3.950	1.240	([8])	([8])	.590	5.780	.630	--------	([9])	--------	.630	10.90	.810	--------	--------	.282
August 31, 1933	[4] 474,500	3.390	.970	([8])	.080	.560	5.000	.620	--------	([9])	--------	.620	12.40	.788	.030	--------	.226
August 31, 1935	[4] 424,248	3.260	1.023	([8])	.081	.411	4.775	.131	--------	([9])	.184	.315	6.60	.659	.080	--------	.310
August 31, 1938	[4] 392,136	3.592	.977	([8])	.095	.368	5.032	.133	--------	([9])	.229	.362	7.19	.684	.037	--------	.185
Siscoe, Quebec, 1937	[4] 200,502	3.285	1.047	.063	.062	.350	4.807	.715	--------	.028	.444	1.187	24.69	1.001	--------	$0.500	--------
Dome, Ontario:																	
1928	[4] 548,000	2.394	1.010	.035	.042	.176	3.657	.813	--------	.111	.126	1.050	28.71	.173	--------	--------	.420
1933	[4] 546,500	2.104	1.144	.035	([8])	.197	3.480	.366	--------	.191	.179	.736	21.15	.957	.040	--------	.265
1937	[4] 576,300	3.107	1.110	.035	([8])	.255	4.507	.337	--------	.342	.338	1.017	22.56	.966	.007	--------	.654
Lake Shore, Ontario:																	
1931	[4] 698,624	3.851	.990	([8])	([8])	.196	5.037	.870	--------	([9])	.123	.993	19.71	.738	--------	--------	--------
1935	[4] 833,094	3.914	1.264	([8])	([8])	.132	5.310	1.053	--------	1.737	.189	2.979	56.10	1.248	--------	--------	.189
1937	[4] 879,559	4.737	1.072	([8])	([8])	.142	5.951	.339	--------	([9])	.331	.670	11.26	1.850	--------	--------	.067
Alaska-Juneau, Alaska:																	
1930	[10] 3,924,460	[11] .287	.227	([8])	([8])	[11] .063	[11] .577	[11] .053	[11] 0.013	--------	[11] .006	[11] .072	12.48	.013	[11] .031	--------	--------
1937	[10] 4,442,760	[11] .347	.245	([8])	([8])	[11] .057	[11] .649	[11] .024	--------	[11] .046	[11] .013	[11] .083	12.80	.051	([11])	[11] .039	[11] .012
Yuba Consolidated Goldfields (dredging), Calif., 1936	[12] 35,969,934	[13] .0449	([8])	([8])	([8])	--------	[13] .0449	[14] .0186	--------	([9])	--------	[13] .0186	41.43	.0076	[13] .0015	--------	[13] .0032
Natomas dredging, Calif.:																	
1935	[12] 21,756,593	[13] [15] .0341	([8])	([8])	([8])	([8])	[13] [15] .0341	--------	--------	.0055	.0044	.0099	29.03	.0061	--------	[13] .0078	[13] .0053
1937	[12] 18,840,213	[13] [15] .0465	([8])	([8])	([8])	([8])	[13] [15] .0465	--------	--------	.0084	.0053	.0137	29.46	.0062	--------	.0021	.0133
Granite gold mining, Mont., 1931	14,222	7.175	--------	([8])	.213	([8])	7.388	.615	--------	.442	.352	1.409	19.07	--------	.152	4.083	.171
North Star, Calif., 1928	[16] 108,600	4.920 [17] 1.360	1.130	([8])	.180	.210	7.800	--------	--------	.180	.140	.320	4.10	--------	--------	.090	.053
Noranda, Quebec, 1937	2,024,468	--------	--------	--------	--------	--------	3.892	.255	--------	([9])	.138	.393	10.10	.922	.032	--------	--------
Consolidated Coppermines, Nev., 1937	1,281,205	1.082	[18] 1.126	--------	--------	--------	2.208	.105	.003	([8])	.124	.232	10.50	--------	--------	--------	.224

Engels, Calif., 1929	395,042	1.049	.653 [19] 1.500	.087	(8)	.024	1.813	.206	.067	.063	.162	.498	27.46			1.436	.262
Miami, Ariz.:																	
1929	[4] 5,017,983	.392	.359 [19] .390	.021	.010	.071	.853	.085		.081	.053	.219	25.68			.160	
1937	4,602,205						1.152 [19] .365	.057	.007	[20] .136	.029	.229	19.88	.005		.070	.014
Utah Copper, Utah 1927	13,811,500	[21] .469	.452 [19] .435	(8)	(8)	(8)	.921	.090		(9)	.021	.171	18.57	.006		.017	.236
Hecla, Idaho:																	
1930	310,876	3.102	.700	.142	(8)		4.723	.393		[22] .779		1.172	24.81	.373			
1937	289,074						4.111	.441		.484	.408	1.333	32.43	.751			[23] .376
Tintic Standard, Utah:																	
1930	96,173						8.686	.375		1.014	1.092	2.481	28.56	.586	.670		3.276
1937	79,285		[19] 12.873				8.675	.155		1.000	1.055	2.210	25.47	.995			[23] .577
Eureka Standard, Utah, 1937	35,847						12.423 [19] 8.575	.571		(9)	.755	1.326	10.67	1.523			.093
Park Utah, Utah:																	
1930	171,750	7.689	.391	(8)	(8)	.243	8.323	.612		.338	.408	1.358	16.32	.00			.546
1937	93,014						9.335	.763	(8)	(8)	[24] 1.186	1.949	20.88				.477
Utah-Apex, Utah, year ended Aug. 31, 1929	197,465	[25] 5.973	1.175	(8)	(8)	.008	[25] 7.156	.252		.112	1.027	1.391	19.44				.277
Silver King Coalition, Utah:																	
1930	155,950						[26] 11.743	.472		.310	.369	1.151	9.80	.190			.393
1933	101,543						[27] 9.640	.535		.694	.405	1.634	16.95	.292			.080
1937	141,280						[28] 10.665	.497		.801	.361	1.659	15.56	1.966			.120
Minnesota iron mines,[29] 1929:																	
Open pit	37,862,622						[30] .496			[31] .471	[32] .569	1.040	209.70				
Underground	9,060,289						1.333			[31] .471	[33] .636	1.107	83.05				

[1] Taken from published annual reports, unless otherwise noted.
[2] Including exploration and development.
[3] Exploration and development of other properties.
[4] Per ton milled.
[5] Includes $0.22 silicosis assessment.
[6] Includes $0.071 silicosis assessment.
[7] Includes $0.121 silicosis assessment.
[8] Included in other items.
[9] Included with income tax.
[10] Tons trammed.
[11] Per ton trammed.
[12] Cubic yards.
[13] Per cubic yard.
[14] Includes depletion.
[15] Includes depreciation.
[16] Per ton of ore crushed.
[17] Development outlay.
[18] Freight and treatment charges; no milling.
[19] Freight and treatment charges.
[20] Includes "other expenses at mine."
[21] Includes ore delivery to concentrator, $0.0926.
[22] Includes some miscellaneous mine expense.
[23] Net.
[24] Taxes, general and administrative expense, fire insurance, and stock-transfer expense.
[25] After crediting lessees royalties, $0.127.
[26] Including $0.579 compensation-insurance reserve.
[27] Including $0.948 compensation-insurance reserve.
[28] Including $0.515 compensation-insurance reserve.
[29] Craig, John J., Mining Directory of Minnesota: Bull., Univ. of Minn., vol. 40, No. 39, May 1, 1937, p. 226.
[30] Not including beneficiation.
[31] Average for open-pit and underground ore.
[32] Includes royalty, $0.456, and depreciation.
[33] Includes royalty, $0.447, and depreciation.

TABLE 61.—*Total costs of producing ore, concentrates, or bullion*—Continued

Company and year	Tons mined or milled	Operating costs per ton mined or milled						Overhead costs per ton						Income tax	Amortization items		
		Mining	Milling	General			Total	Depreciation	Interest paid	Taxes (except income taxes)	Other	Total			Exploration	Depletion	Interest received and other income credit
				Surface	Compensation	Other						Amount	Percent of operating cost				
Minnesota iron mines,[29] 1935:																	
Open pit	16,144,548	--------	--------	--------	--------	--------	$0.512	--------	--------	[31]$0.962	[34]$0.629	$1.591	253.00	--------	--------	--------	--------
Underground	3,792,734	--------	--------	--------	--------	--------	1.257	--------	--------	[31].962	[35].638	1.620	128.8	--------	--------	--------	--------
Michigan iron mines, 1936 [36]:																	
A. Marquette range	4,627,889	$1.374	--------	(8)	(8)	[37]$0.034	1.408	$0.084	$0.003	[31].168	[38].437	[39].693	49.22	--------	--------	--------	--------
B. Gogebic range	3,710,468	1.787	--------	(8)	(8)	[37].033	1.820	.110	.001	[31].292	[38].513	[40].916	50.31	--------	--------	--------	--------
C. Menominee range	2,163,679	1.533	--------	(8)	(8)	[37].037	1.570	.158	.003	[31].132	[35].393	[41].686	43.69	--------	--------	--------	--------

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[29] Craig, John J., Mining Directory of Minnesota: Bull., Univ. of Minn., vol. 40, No. 39, May 1, 1937, p. 226.

[31] Average for open-pit and underground ore.

[34] Includes royalty, $0.457, and depreciation.

[35] Includes royalty, $0.389, and depreciation.

[36] Pardee, F. G., and Eddy, G. E., General Statistics Covering Costs and Production of Iron Mines: Dept. of Conservation, Geol. Surv. Div., 1937, 8 pp.

[37] Superintendence.

[38] General overhead, marketing, and royalty.

[39] Rail and boat freight and cargo insurance, $1.5017 additional.

[40] Rail and boat freight and cargo insurance, $1.7239 additional.

[41] Rail and boat freight and cargo insurance, $1.6715 additional.

REALIZATION FROM SALE OF ORE, CONCENTRATES, AND BULLION

With the exception of gold, metal prices are subject to frequent fluctuation, resulting in corresponding variations in the value of their ores.

Metal prices are quoted periodically in technical and trade journals and average yearly prices in the principal markets are discussed in the Minerals Yearbook, published annually by the Bureau of Mines. From time to time charts have been prepared showing price trends of the principal metals over long periods; a recent one was published by the Engineering and Mining Journal.[99]

Prices are quoted delivered f. o. b. at certain basing or delivery points such as New York for copper, New York and St. Louis for lead, and St. Louis for zinc. For several years the Federal Government has established prices for silver produced from domestic ores, which have been considerably above the world market price. Lake Superior iron ores are quoted f. o. b. lower lake ports at prices per unit of contained iron that vary with the class of ore and the total iron content.

A discussion of metal prices and the marketing of metals and ores is beyond the scope of this bulletin, and the subject is mentioned merely to introduce the discussion that follows. For information on marketing the reader is referred to the work of Spurr and Wormser,[1] to which 30 well-known specialists contributed.

Although the quoted prices are used as a basis for computing the value at the mine or mill of nonferrous metals contained in the ores or concentrates, the mine value is less than quoted value by charges for freight on ores or concentrates to the smelting plants, smelting charges, and smelter deductions for losses in treatment and for deleterious constituents of the ore, cost of shipping unrefined metal to refineries, and the refining charges, marketing costs, and the like.

From the value of Lake Superior iron ores quoted delivered at lower lake ports must be deducted cost of rail freight from the mines to upper lake ports, handling and loading charges at the docks, lake freight, cargo insurance, unloading cost and sampling charges at lower lake ports, and selling costs to arrive at the net value at the mines (see footnotes 39, 40, 41, table 61).

Table 56, included under the discussion of ore dressing, gave examples of open schedules covering gold and silver ores at western (United States) lead smelters. Table 62[2] gives similar schedules at copper smelters.

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[99] Statistical and Graphical Record, Eng. and Min. Jour. Metal Prices, Supplement to Eng. and Min. Jour., February 1938.

[1] Spurr, Josiah Edward, and Wormser, Felix, The Marketing of Metals and Minerals: McGraw-Hill Book Co., New York, 647 pp.

[2] Gardner, E. D., and Allsman, Paul T., Open Schedules for Gold and Silver Ores and Concentrates at Western Custom Smelters: Bureau of Mines Inf. Circ. 6926, 1936, pp. 13-14.

TABLE 62.—*Schedules for gold and silver ores at copper smelters (July 1936)*

Plant	Payments											
	Gold [1]			Silver [2]			Copper [3]			Lead [4]		
	Minimum paid for, ounces per ton	Payments		Minimum paid for, ounces per ton	Minimum deduction, ounces per ton	Percent paid for	Deductions		Percent of quotation after deduction	Deductions		Percent of quotations after deduction
		Classes of ore, ounces per ton	Rate per ounce				Pounds per ton [5]	Cents per pound		Units	Cents per pound	
El Paso, Tex.	0.03	All	$32.31825	------	0.5	[6] 95	8	2.525	[7] 95	1.5	3.0	[8] 60
Garfield, Utah [9]	.03	0 to 3 Over 3	31.81825 32.31825	1.0	.5	95	15	2.5	100	0	3.5	[10] 50
Midvale, Utah	.02	0 to 5 Over 5	31.81825 (11)	1.0	.5	95	[12] 15	3.0	90	----	------	------
Anaconda, Mont.	.01	All	31.81825	------	1.0	[13] 95	[12] 10	2.5	90	----	------	0
Tooele, Utah [14]	.02	All	31.85	1.0	------	95	15	2.775	100	----	3.525	[10] 50
Do.[15]	.02	0 to 3 3 to 6 6 to 10 Over 10	31.85 32.20 32.90 33.25	1.0	------	95	15	2.775	100	----	3.525	[10] 50
Miami, Ariz.	.03	All	32.20	1.0	------	95	[16] 10	2.5	90	----	------	0
Superior, Ariz.[17]	.03	All	32.20	1.0	------	95	[18] 8	2.5	90	----	------	0
Do.[19]	.03	All	32.20	1.0	------	95	[18] 8	2.5	90	----	------	0

[1] Payments on gold are based on the new mint price of $35 per ounce (net $34.9125).

[2] Payments on silver are based on the new mint price for American mined ore at $0.77 per ounce.

[3] Payments on copper are based on the Engineering and Mining Journal quotations.

[4] Payments on lead are based on the New York quotation for common desilverized lead.

[5] When assay is less than the deduction, the shipper is charged with the deficiency, except as further noted.

[6] Less a deduction of 1½ cents per ounce.

[7] Nothing paid for copper less than ½ percent.

[8] Nothing paid for lead less than 5 percent by wet assay.

[9] Siliceous ore open schedule.

[10] Nothing paid for lead less than 2 percent by wet assay.

[11] On direct smelting ores containing over 5 ounces gold per ton, the price per ounce for the excess will be to the mutual agreement between the buyer and shipper.

[12] Minimum deduction.

[13] Less a deduction of 2 cents per ounce.

[14] Dry-ore rate.

[15] Gold-concentrate rate.

[16] Minimum deduction 10 pounds and maximum deduction 20 pounds per ton. No payment or deduction will be made when copper is less than 0.8 percent.

[17] Crude-ore open schedule for carload lots.

[18] Minimum deduction 8 pounds per ton and maximum deduction 20 pounds per ton.

[19] Concentrate open schedule for carload lots.

TABLE 62.—*Schedules for gold and silver ores at copper smelters (July 1936)*—Con.

Plant	Deductions										
	Treatment charge				Penalties						
	Base		Add for excess gross value over base percent	Maximum	Zinc		Arsenic (As), antimony (Sb), and tin (Sn)			Bismuth	
	Per ton	Gross value			Units free	Charge per unit for excess	Element or combination	Unit free	Charge per unit for excess	Units free	Charge per unit for excess
El Paso, Tex.	$3.50	0 to $20 Over $20	10	$6.00	5	$0.30	Arsenic Sb+Sn	2 1	$0.50 1.50	0.1	$0.50
Garfield, Utah.[9]	3.50	$0 to $20 Over $20	10	5.50	6	.30	As+Sb	2	.50	------	0
Midvale, Utah.	5.00	$0 to $30 Over $30	10	7.00	6	.30	----------	------	0	------	0
Anaconda, Mont.	4.00	$0 to $15 Over $15	10	[20] 5.50	------	0	----------	------	0	------	0
Tooele, Utah,[14]	4.00	0 to $25 Over $25	10	6.00	------	0	----------	------	0	------	0
Do.[15]	4.00	$0 to $25 Over $25	10	6.00	------	0	----------	------	0	------	0
Miami, Ariz.	3.50 6.00	$0 to $15 $15 to $40 $40 to $100 Over $100	10 5	6.00 8.00	----	0	----------	------	0	------	0
Superior, Ariz.[17]	3.50	$0 to $15 $15 to $100 Over $100	10 5	6.00 15.00	----	0	----------	------	0	------	0
Do.[19]	2.50	$0 to $15 $15 to $100 Over $100	10 4	5.00 -------	-----	0	----------	------	0	------	0

[9] Siliceous ore open schedule.
[14] Dry-ore rate.
[15] Gold-concentrate rate.
[17] Crude-ore open schedule for carload lots.
[19] Concentrate open schedule for carload lots.
[20] To the base treatment charge also add 12 cents for each unit of iron and deduct 2.5 cents for each unit of silica in excess of alumina; treatment charge is not to exceed maximum.

It is noted that no payment is made for zinc in lead ores and concentrates but that, on the contrary, there is a penalty for zinc in excess of specified maximum percentages. On the other hand, ordinarily no payment is made for small amounts of lead in zinc ores and concentrates; for larger percentages of lead most schedules include payment for part of the lead at a price considerably below the quoted price. Where zinc occurs in copper ores or concentrates in excess of certain maximum percentages, copper smelters usually charge extra for smelting, which reduces the net value of the copper to the shipper. Zinc smelters usually pay for part of the copper if in excess of stipulated minimum percentages but at a price considerably below the current quotation.

To illustrate the effect of the factors mentioned in a general way above, the following examples of schedules and calculations from Information Circular 6926 are given:

EXAMPLE 1:

SILICEOUS ORE SCHEDULE EFFECTIVE *July 7, 1936*

To: ________________________________

Delivery: F. o. b. ____________________ up to 100 tons per month; excess, buyer's option.

PAYMENTS FOR METALS

Gold.—If 0.03 ounce per dry ton and up to and including 3.0 ounces, pay for all at $19.00 per troy ounce; if over 3.0 ounces, pay for all at $19.50 per troy ounce, plus 90 percent of realized gold premium in excess of $20.67 per troy ounce. Under present Government price this is equivalent to paying for 100 percent at $31.81825 per ounce when under 3.0 ounces and $32.31825 per ounce when over 3.0 ounces gold content per ton.

Silver.—If 1.0 troy ounce per dry ton or over, pay for 95 percent at the applicable Handy & Harman quotation, or at the present realized Mint price of 77 cents per ounce, provided silver qualifies for Government purchase and affidavit is furnished; minimum deduction 0.5 ounce.

Copper.—Deduct 0.75 unit from wet copper assay and pay for remaining copper at prevailing electrolytic cathode quotation, less a deduction of 2.5 cents per pound of copper paid for.

Lead.—Pay for 50 percent of wet assay less a deduction of 3.5 cents per pound at ———— prevailing lead quotation. No lead paid for if less than 2.0 percent by wet assay.

Quotations.—Silver, lead, and copper quotations are based on average for calendar week including date of arrival of last car of each lot at plant of buyer, and on a bullion freight rate of $12.50 per ton to New York City.

No payment will be made for any metal or content except as above specified.

DEDUCTIONS

Base charge.—$3.50 per net dry ton on ores having a gross value of $20.00 or less. Increase base charge 10 percent of excess gross value above $20.00 up to a base charge of $5.50 per ton.

Sampling.—A flat charge of $5.00 will be made for sampling, handling, and assaying each lot containing less than 10 tons dry weight when value per ton is $200.00 or less. When value per ton exceeds $200.00, sampling, handling, and assaying charge will be $7.50 per lot. No charge will be made for sampling, handling, and assaying lots containing 10 tons or over dry weight.

Impurities.—Zinc 6 percent free; charge for excess at 30 cents per unit. Arsenic and antimony combined, 2 percent free; charge for excess at 50 cents per unit.

Freight.—All railroad, freight, and delivery charges are for the account of the shipper. Deduct from settlement, freight, and other advances made by buyer.

Assaying.—Buyer's assays will govern if seller fails to make or submit assays prior to acceptance of payment. Seller should advise at time shipment is made whether or not he desires to submit assays and also direct buyer where to send control pulp samples.

Duration.—Rates on ores not under contract are subject to change 30 days after date shown above. This rate cancels and supersedes all former rates quoted.

EXAMPLE OF CALCULATIONS

The method of calculating returns on a shipment of ore based on the preceding schedule follows:

Wet weight of lot, 36.28 tons.
Moisture, 8.5 percent.
Dry weight of lot, 36.28 (1.00−0.085)=33.20.

Chemical analysis:	*Ounces per ton*
Gold	1.27
Silver	2.10
	Percent
Copper	1.2
Zinc	6.4
Arsenic	.60
Antimony	.72
Bismuth	.08
Iron	12.4
Sulfur	19.1
Silica	49.7
Alumina	6.0
Total	96.20

Payments:	
Gold, 1.27×31.81825	$40. 41
Silver (2.10−0.5)×0.77	1. 23
Copper, market quotation 9.5 cents per pound. 20 (1.2−0.75)×(0.095−0.025)	. 63
Lead	None
Gross payments per dry ton	42. 33
Deductions:	
Treatment charge:	
For $20 gross value, per ton	3. 50
Plus ($42.33−20) 0.1	2. 23
	5. 73
Maximum	5. 50
Penalties:	
Zinc (6.4−6)×0.30	12
Arsenic plus antimony (0.60+0.72)−2.0	None
Total deduction per dry ton	5. 62
Net payments per dry ton	36. 71
Total returned by the smelter to the shipper 33.20×36.71	1, 218. 77

In this example the gross value of the ore per dry ton based on full quoted market prices of the contained gold, silver, and copper would have been as follows:

Gold, 1.27 ounces @ $35	$44. 450
Silver, 2.10 ounces @ $0.77	1. 617
Copper, 1.2 percent of 2,000 pounds equals 24 pounds @ $0.095	2. 280
Total per dry ton	48. 347

After deductions are made for metal losses, this figure becomes $42.33, as shown; and after smelter charges and penalty deductions it becomes $36.71 per ton, 75.9 percent of the gross value (not including zinc). Assuming that the smelter is some distance from the mine, a further deduction would be necessary to cover freight.

EXAMPLE 2

DOMESTIC LEAD ORES AND CONCENTRATES

The following terms are subject to the general clauses and are subject to prompt acceptance. Unless shipments are begun within 30 days this quotation is canceled automatically.

PAYMENTS

Gold.—If 0.03 ounce per dry ton or over, pay for all at the rate of $20 per troy ounce plus 90 percent of the realized gold premium in excess of $20.67 per troy ounce. Under present Government net realized price ($34.9125), this is equivalent to $32.81825 per troy ounce.

Silver.—Pay for 95 percent (minimum deduction of ½ ounce) at the average Handy and Harman New York silver quotations for the calendar week, including date of arrival of last car of each lot at plant of buyer; or, if higher, at the realized Mint price, provided silver qualifies for Government purchase and affidavit is furnished, less a deduction in either case of 1½ cents per ounce. Present Mint price is 77 cents per ounce.

Lead.—Deduct from the wet-lead assay 1.5 units and pay for 90 percent of the remaining lead at smelter quotation less 1.425 cents per pound of lead accounted for. Nothing paid for lead if less than 5 percent by wet assay.

Copper.—Deduct from the wet-copper assay 8 pounds and pay for 95 percent of the remaining copper at the daily net refinery quotations for electrolytic cathodes as published in the Engineering and Mining Journal of New York averaged for the calendar week, including date of arrival of last car of each lot at the plant of the buyer less a deduction of 5.025 cents per pound of copper accounted for. Nothing paid for copper if less than ½ percent by wet assay. No payment will be made for any metal or content except as above specified.

DEDUCTIONS

Base charge.—$3.70 per net dry ton of 2,000 pounds; provided the sum of payments for gold, silver, lead, and copper does not exceed $25 per ton. Add to the base charge 10 percent of the excess over $25 to a maximum charge of $6.70 per ton.

Insoluble.—Allow no units free; charge for all at 5 cents per unit, fractions in proportion.

Zinc.—Allow 5 units free; charge for the excess at 30 cents per unit, fractions in proportion.

Sulfur.—Allow 2 units free; charge for the excess at 20 cents per unit, fractions in proportion. Maximum charge $2 per dry ton of material.

Arsenic.—Allow 2 units free; charge for the excess at 50 cents per unit, fractions in proportion.

Antimony and tin (combined).—Allow 1 unit free; charge for the excess at $1.50 per unit, fractions in proportion.

Bismuth.—One-tenth of 1 percent (0.1 percent) of the lead content by wet assay will be allowed free. The excess will be charged at 50 cents per pound, fractions in proportion.

Moisture.—A minimum deduction of 1 percent will be made from wet weight; when over 1 percent is contained, the actual moisture will be deducted.

Delivery.—F. o. b. unloading bins.

Taxes.—Deduct Federal or State taxes, import duties, stamps, and/or other charges now or hereafter imposed..

Freight.—All railroad freight and delivery charges for account of shipper. Deduct from settlement freight and other advances made by buyer.

GENERAL CLAUSES GOVERNING ALL OPEN SCHEDULES

1. Weighing, moisture, and ore sampling (at which seller or a representative may be present), as done by buyer according to standard practice, promptly after receipt of product, will be accepted as final. The absence of seller or a representative shall be deemed a waiver of the right in each instance. After sampling, the product may be placed in process, commingled, or otherwise disposed of by buyer. In case of disagreement on assays, an umpire shall be selected in rotation from a list mutually agreed upon, whose assays shall be final if within the limits of the assays of the two parties; and if not, the assay of the party nearer to the umpire shall prevail. Losing party shall pay cost of umpire. In case of seller's failure to make or submit assays, buyer's assays shall govern.
2. All schedules on ore not under contract for a definite period of time are subject to change without notice.
3. The rates quoted herein are for carload lots. On any lot containing 1 ton or less there will be a sampling and handling charge of $10. This charge will be decreased by $1 for each ton in excess of 1 ton.
4. The rates quoted apply only to ore in bulk. Fifty cents per ton additional will be charged for ore in sacks to cover extra cost of handling.
5. In the event quotation date should fall on a legal holiday or one upon which no quotation is issued, the next succeeding quotation will be used in settlement.
6. In this schedule where the word "ton" is used it is understood to be a ton of 2,000 pounds avoirdupois; where the word "ounce" is used, as referring to gold and silver, it is understood to mean the troy ounce; and where the word "unit" is used it is understood to mean 1 percent of a ton, or 20 pounds avoirdupois.
7. In order that delivery of the ore at our plant may not be delayed unnecessarily we make it a general rule that *the freight charges must be prepaid or guaranteed by the shipper*.
8. The rates quoted herein are based on present existing scale for common labor at the smelting works and present published all-rail freight rates on lead and copper bullion from smelter to New York City. Any increase or decrease is for account of seller, and proper deduction or credit shall be made accordingly.

EXAMPLE OF CALCULATIONS

The method of calculating returns based on the foregoing schedule follows:

Wet weight of lot, 137.2 tons.
Moisture, 3.0 percent.
Dry weight of lot, 137.2 (1.00−0.03)=133.1 tons

Chemical analysis:	*Ounces per ton*
Gold	0. 53
Silver	1. 65
	Percent
Lead	2. 6
Copper	. 82
Zinc	6. 7
Arsenic	2. 4
Antimony	1. 4
Tin	0
Bismuth	. 08
Iron	15. 0
Sulfur	19. 7
Insoluble	28. 0
Lime	5. 6
Magnesia	4. 8
Loss on ignition	9. 6
Total	96. 7
Payments:	
Gold, 0.53×32.81825	$17. 39
Silver, 1.65×.95 (0.77−0.015)	1. 18
Lead, less than 5 percent	None
Copper, quotation 9.5 cents per pound 0.95 [(0.82 x 20)−8]×(0.095−0.05025)	. 36
Gross payments per dry ton	$18. 93
Deductions:	
Treatment charge: On ore between $0 and $25 gross value	3. 70
Penalties:	
Insoluble 28.0×0.05	1. 40
Zinc (6.7−5)×0.30	. 51
Sulfur (19.7−2)×0.20=$3.54, maximum	2. 00
Arsenic (2.4−2)×0.50	. 20
Antimony and tin, (1.4−1)×1.50	. 60
Bismuth [0.08−(0.001×2.6)] 20×0.50	. 77
Total deductions per dry ton	9. 18
Net payments per dry ton	9. 75
Total returned by the smelter to the shipper 133.1×$9.75	1, 297. 73

In this example, the gross value per dry ton of the contained gold, silver, and copper only, based upon full quoted market prices, would have been as follows:

Gold, 0.53 ounce at $35	$18. 55
Silver, 1.65 ounces at $0.77	1. 27
Copper, 0.82 percent or 17.4 pounds at $0.095	1. 65
	21. 47

After deductions for metal losses, this figure becomes $18.93 as shown, and after treatment charge and penalties is further reduced to $9.75 per dry ton, which is only 45.4 percent of the gross value of the gold, silver, and copper. Net value at the mine would be less than this by the cost of shipping to the smelter.

It is common practice for the smelter to pay the cost of freight and deduct it from the payment made to the shipper.

To further show the relation of gross value to net returns, the data in table 63 are presented. These data are taken from actual returns made to shippers, and bases of payment, deductions, and charges are given in the references (see column 1).

It is noted that for gold and silver ores shipped in the crude state or in the form of concentrate (first six mines listed), the net return to the shipper varies from about 54 to 84 percent of gross value at full quoted prices; whereas at the Vulture mine, where the gold and silver are shipped as high-grade precipitate, the net return is 92.7 percent. Where the precious metals are shipped in the form of semirefined bullion the return is often even higher. It is, therefore, decidedly to the advantage of the shipper to reduce gold and silver to bullion if his ore is amenable to treatment by amalgamation or cyanidation. In some instances recovery of part of the gold and silver by these processes and of the remainder in concentrates may be the most economical practice.

In general, the data in table 63 serve to emphasize the point previously discussed under "Ore dressing" with respect to the advantages of concentrating the valuable metals within the smallest bulk possible commensurate with good recovery to obtain the highest net return on the contained metal value.

TABLE 63.—*Comparison of gross and net values per dry ton of ores and concentrates*

Mine and reference	Year	Settlement assays					Gross value based on full quoted prices [1]		Gross value per ton paid	Deductions				Net value at mine or mill
		Ounces per ton		Percent										
		Gold	Silver	Copper	Lead	Zinc	Metals paid for	Gross value per ton		Freight per dry ton	Treatment	Penalties	Total	
Red Cloud, Nev.[2]	1936	0.815	0.100				Gold	[3] $28.45	$25.93	$4.61 [4] 2.04	$3.25	$0.87	$10.77	$15.16
Lake Shore, Nev.[2]	1936	1.552	.775				do	[3] 54.20	49.40	7.19	5.00	[5] .23	12.42	36.98
Keystone-Barefoot, Nev.[2]	1936	1.46		0.52			do	[3] 50.97	46.45	5.50	4.60		10.10	36.35
Boss, Nev.[2]	1936	.67	1.80	2.02			Gold, silver, copper	[3] 28.88	24.96	4.69	3.84		8.53	16.43
Blossom, Nev.[2]	1936	3.845	47.62				Gold, silver	[3] 170.91	159.10	12.32	7.70	[5] .89	20.91	138.19
Ashby, Nev.[6]	1935	6.53	5.0	.30	0.60		do	[3] 231.83	215.04	11.55 [4] 1.87	6.20		19.62	195.42
Vulture, Ariz.[7]	1936	[8] 415.40	[8] 795.85				do	[8] 15,167.55	14,502.41	122.50	312.50		435.00	14,067.41
Golden Mile, Nev.[6]	1936	.345	1.15	1.66			Gold, silver, copper	[3] 15.97	13.49	3.75 [4] 2.60	1.35	1.56 [5] .08	9.34	4.15
Randall, Nev.[5]	1935	.750	22.25			0.50	Gold, silver	40.52	37.44	5.02	5.24		10.26	27.18
New Year, Nev.[6]	1936	.420	4.60	.18			do	[3] 18.20	16.52	4.16 [4] 1.35	3.50		9.01	7.51
Evening Star, Nev.[9]	1935	.90	43.0	.20		.20	do	[3] 64.53	60.09	10.64	7.50	[5] 3.82	21.96	38.13
Willard, Nev.[9]	1935	.575	3.4				do	[3] 22.25	20.16	7.29	4.90		12.19	7.97
Kennedy, Nev.[6]	1935	1.0825	22.62	2.65	1.00	.50	Gold, silver, copper	60.18	53.40	5.20	5.55	[5] .73	11.48	41.92
Zenoli, Nev.[10]	1936	.005	21.2	1.11		2.00	Silver, copper	18.35	15.99	4.04	3.50		7.54	8.45
Silver Bell, Nev.[10]	1937		47.0	.95	33.6	9.00	Silver, lead, copper	79.10	60.69	13.09	2.49	3.84	19.42	41.27
Golden Belt, Ariz.[11]	1936	2.48	21.3	.52	10.6		Gold, silver, lead, copper	113.67	102.08	8.57	5.50	[5] .19	14.23	87.85
Silver Cord, Ariz.[11]	1935	.685	21.96	.43	9.5	6.2	Gold, silver, lead	49.34	42.89	4.75	5.49	5.36	15.60	27.29
Moho, Nev.[6]	1935	1.015	6.3		1.4		Gold, silver	[3] 39.48	36.02	4.72 [4] 1.61	5.16		11.49	24.53
Budget, Nev.[2]	1936	1.3875	1.95	.1	3.6		Gold, silver, lead	[3] 54.25	45.66	5.06	5.50		10.56	35.10
Pansy Lee, Nev.[12]	1937	.525	36.25	.05	3.25		do	50.19	44.05	4.67	5.50	[5] .09	10.26	33.79
Eureka-Croesus, Nev.[10]	1936	.40	19.0		7.50	.5	do	[3] 35.80	27.44	6.56	5.22		11.78	15.66
Copper Basin, Nev.[13]	1937	.02	.165	7.54			Gold, copper	18.45	13.23	3.10 [4] 1.03	2.25	[5] .75	7.13	6.10
Do.[13]	1936	.90	4.39	7.675			Gold, silver, copper	48.00	40.07	5.39 [4] 1.84	4.00	[5] .82	12.05	28.02
Buzzard, Nev.[13]	1936	3.255	36.575	.76			do	[3] 143.19	131.96	9.90 [4] 5.50	5.67	[5] 1.71	22.78	109.18
Gold Butte, Nev.[13]	1938	.0775	19.25	.1	3.1		Gold, silver, lead	[3] 20.47	16.94	3.33 [4] 3.08	3.60	[5] .12	10.13	6.81
July, Nev.[10]	1937		.80		9.6	39.8	Lead, zinc	69.20	25.15	6.26	2.88		9.14	16.01

[1] Figured at 100 percent of content of metals paid for.

[2] Vanderburg, W. O., Reconnaissance of Mining Districts in Clark County, Nev.: Bureau of Mines Inf. Circ. 6964, 1937, 81 pp.

[3] Gold at $34.9125 per ounce.

[4] Trucking to railroad.

[5] Assaying and sampling charges.

[6] Vanderburg, W. O., Reconnaissance of Mining Districts in Mineral County, Nev.: Bureau of Mines Inf. Circ. 6941, 1937, 79 pp.

[7] Metzger, O. H., Gold Mining and Milling in the Wickenburg Area, Maricopa and Yavapai Counties, Ariz.: Bureau of Mines Inf. Circ. 6991, 1938, 78 pp.

[8] Cyanide precipitate.

[9] Vanderburg, W. O., Reconnaissance of Mining Districts in Pershing County, Nev.: Bureau of Mines Inf. Circ. 6902, 1936, 57 pp.

[10] Vanderburg, W. O., Reconnaissance of Mining Districts in Eureka County, Nev.: Bureau of Mines Inf. Circ. 7022, 1938, 66 pp.

[11] Guiteras, Jos. R., Gold Mining and Milling in the Black Canyon Area, Yavapai County, Ariz.: Bureau of Mines Inf. Circ. 6905, 1936, 51 pp.

[12] Vanderburg, W. O., Reconnaissance of Mining Districts in Humboldt County, Nev.: Bureau of Mines Inf. Circ. 6995, 1938, 54 pp.

[13] Vanderburg, W. O., Reconnaissance of Mining Districts in Lander County, Nev.: Bureau of Mines Inf. Circ. 7043, 1939, 83 pp.

ENGINEERING VALUATION OF MINING PROPERTIES

Engineering valuations of mining properties are made for a number of purposes, principally the following:

1. For purposes of purchase or sale on behalf of prospective buyers or vendors.
2. For purposes of financing by bond issues or sales of stock.
3. As a basis for taxation.
4. In connection with tresspass or damage suits.

Theoretically the value of a mine should be the same, whatever the purpose of the valuation; as mining operations are conducted for profit, this value at any time is the present worth of the profits that will be earned after amortization of capital investment during the life of the mine.

It is apparent from a consideration of the discussions and data presented in the preceding sections of this bulletin that many variable factors impossible of exact determination affect the profits that may be realized from the exploitation of a given mineral deposit; hence, valuations derived from a considered weighing of all these factors cannot be precise.

Because of these inherent inaccuracies, a valuation is likely to be colored, to some extent at least, by the purpose for which it was made or by the viewpoint of the engineer, no matter how great his moral and intellectual honesty. Careful and conscientious valuations of the same mine made independently by different engineers may differ widely, depending upon the different interpretations of and the weights given to the various engineering and economic factors involved. It is only human for a prudent prospective purchaser to foresee and give more weight to possible contingencies that may arise to increase costs or to reduce income than would the seller who is trying to obtain the maximum possible price for his property.

In spite of the difficulty of accurately estimating future costs and revenues, valuations based on present worth of anticipated profits are most logical and long have been accepted for all the above purposes.

In transactions involving the sale of a mine the price actually paid usually will be decided by negotiation and may differ considerably from that derived by engineering appraisal. In such instances, however, the engineering appraisal is indispensable as a basis for intelligent negotiation.

The subject of mine valuation and the mathematics involved have been discussed at length in a number of standard works, among the best known of which are the following:

Hoskold, H. D., Engineer's Valuing Assistant: Longmans, Green Co., London, New York, and Bombay, 1905, 185 pp.

Hoover, Herbert C., Principles of Mining: McGraw-Hill Book Co., New York, 1909, 199 pp.

Hoover, T. J., Economics of Mining: Stanford University Press, Stanford, Calif., 1933, 547 pp.

Burnham, M. H., Modern Mine Valuation: J. B. Lippincott, Philadelphia, 1912.

Louis, Henry, Mine Valuation: Charles Griffin & Co., London, 1933, 281 pp.

Baxter, C. H., and Parks, R. D., Mine Examination and Valuation: Michigan College of Mining and Technology, Houghton, Mich., 1933, 316 pp.

Finlay, James Ralph, The Cost of Mining: McGraw-Hill Book Co., New York, 1909, 415 pp.

Leith, C. K., Mineral Valuations of the Future: Am. Inst. Min. and Met. Eng., 1938, 116 pp.

Hoskold first prepared and published in 1877 full sets of tables required for computing present worth, and later writers on the subject have reproduced these tables in part. By their use, computations are greatly simplified. They cover a range of dividend and interest rates and terms of years especially suited to mine valuation in which, on account of the risks perculiar to mining, the investor demands a higher return on his money than he would require from other types of productive enterprise where the operation is not dependent on a wasting raw-material resource and the risks are smaller.

Of especial value are tables 6, 7, 8, and 9, in which the factors have been calculated so as to combine income rates of 3½ to 25 percent per annum with interest rates on invested sinking fund of 2½, 3, 3½, and 4 percent, respectively; and tables 10 and 11, in which similar combined factors are given for computing present worth where income is deferred for 1, 2, 3, and 4 years, respectively.

Hoskold and other writers stress the importance of allowing for amortization of capital investment in view of the limited life of a mining operation (determined by the total ore reserves and the rate of production).

The following discussion of mine valuation is confined to brief observations on the engineering principles and data presented in the preceding sections of this bulletin as they affect the use of accepted methods of mine appraisal.

A rational appraisal of the potentialities of a mine for earning profits over a period of years requires a broad understanding of the geology and of the operating and economic problems involved, together with sound judgment. Experienced mining engineers are assumed to be already conversant with most of the subjects previously discussed in this bulletin, but it is hoped that such may find some of the engineering and cost data useful. On the other hand, students and younger practicing engineers may find the discussions useful also in approaching mine-appraisal problems.

Metalliferous deposits may be divided roughly into two classes for valuation purposes:

1. Deposits with substantial tonnages of assured (proven) ore.
    - (*a*) Extensive deposits of uniform tenor (usually low-grade).
    - (*b*) Large ore bodies of erratic occurrence (erratic in grade or of complex mineral combination, or both).
2. Deposits with little or no proven ore.

A further distinction may be made, depending on whether the deposits are supporting established operating mines, that is, have an operating history, or have not yet entered the regular productive stage.

PROPERTIES WITH LARGE ASSURED ORE RESERVES

The number of variable factors in valuations of deposits of type 1–*a* are a minimum. The "porphyry" copper deposits of the southwest, the lead mines of the Southeast Missouri district, and the iron mines of the Lake Superior and Birmingham districts are examples of this type of deposit. Here, substantial reserves will have been proved usually by drilling or underground development before production begins. Hence, the mining and milling operations may be highly systematized and approach in effect the operation of a manufacturing plant, the main difference being that the raw material of a mine—its ore reserve—is definitely limited and not replaceable and, with the exhaustion of the ore, production ceases.

For such deposits the ore-reserve tonnage and grade may be determined by procedures discussed in the sections on sampling and estimation of ore reserves, with a relatively high degree of accuracy and an average yearly rate of production and life can then be predicted with some confidence, although probable fluctuations in price and market demand must be considered. Production costs can be estimated closely, particularly if the mine under consideration has a production record from which accurate cost data can be derived.

The most uncertain elements in valuing deposits of this class are the price that will be received for the product and the rate of production that market demands will support.

Valuation engineers are accustomed to employ curves showing variations in and general trends of metal prices and consumption over a long period of years for projecting prices into the future. Studies of price trends should be supplemented by consideration of the possible effect of such factors as general exhaustion of other mines in the country and the prospects for important new domestic ore discoveries, possible tariff changes, foreign ore discoveries, and competition from secondary or scrap metals and from new synthetic substitutes for metals.

It is therefore apparent that long-range predictions of metal prices are precarious and that mineral economics and the world situation with regard to the reserves of metallic ores in the ground must be considered.

The difficulty in long-range price prediction may not be a serious matter, however, if the nearer range prices can be approximated closely, because, as pointed out by standard works on mine valuation, the present worth of profits that will not accrue for 20 years or more is small. Hotchkiss and Parks [3] have shown that "present value of an annuity of one dollar, using interest rates of 8 percent for return on capital and 4 percent for accrual on redemption fund extended to infinite life at these rates, is $12.50, or 12½ times the yearly income; that of this ultimate maximum present value, 50 percent is attributable to income of the first 10 years, 80 percent to the income of the first 28 years, and 88 percent to the income of the first 40 years." On these assumptions the present worth of income of the first 10 years would be 62.5 percent of that for the first 28 years, and that of the first 28 years about 91 percent of that of the first 40 years. Therefore, it may be concluded that small errors in estimating prices after 15 or 20 years will have little effect upon the present worth and larger errors will have small relative effect.

In general, application of the present-worth method to valuation of mines of type 1, and especially of type 1–*a*, may be made with considerable confidence in the resulting figure, which in most instances probably will be nearer the actual sale price of such mines than it would be for mines of the other types. Hence, a lower rate of return on capital may be used in calculating present worth of type 1 mines than when applying this method to mines where the risk is greater because a greater number of factors are variable and impossible of close determination.

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[3] Hotchkiss, W. O., and Parks, R. D., Total Profit vs. Present Value in Mining: Am. Inst. Min and Met. Eng. Tech. Pub. 708, Class A., Metal Mining No. 66, 1936, 9 pp.

In 1911 the Michigan State Tax Commission adopted the present worth method for valuing iron and copper mines, employing a 6-percent return on capital and a 6-percent sinking fund rate.

The uncertainties in valuations of properties of type 1–*a* apply equally to those where there is less certainty as to production costs and ore reserves, except that for short-life properties prices sometimes may be predicted more accurately. The importance of skilled management in the earning of profits from operation of mines of any class cannot be overemphasized, and therefore, wherever possible the type of management that is likely to be had should be considered.

Additional uncertainties arise in valuing properties in the other classes. With deposits of type 1–*b*, there is more question as to operating costs that will be realized and as to the grade of ore that will be mined, these factors often being complementary. These deposits are also typically irregular with respect to characteristics that determine the mining methods and practices that can be employed, and with increased depth the mineralogical character of the ore may change, requiring changes in ore-dressing treatment. The effect upon costs of natural mining conditions and the methods of mining employed has been discussed under Stoping.

The mining method adopted may depend to a considerable extent on the policy with respect to grade of ore that is to be mined.

Thus, if it is decided to mine selectively only the better-grade ore, leaving in the mine that upon which little profit can be expected, a relatively high-cost mining method may be required. Or, if the policy is to mine this ore by a method that will not preclude mining the lower-grade ores later when market prices are high and demand heavy, a still more expensive mining system may be necessary. Obviously, a policy of mining only the better-grade ore will result in smaller ore reserves and a shorter productive life. Normally this will result in a higher valuation for this class of deposit, because a high income rate is realized during early years when present-worth discounts are a minimum, unless the life were so shortened that high amortization charges would overbalance the greater profits.

It has been argued [4] that since the primary object of mining is for profit, the operating policy that will result in the highest present worth of profits is the one that should be adopted. Even aside from the question of conservation of natural mineral resources, however, this is not invariably so. Thus, in the case of integrated companies that are engaged in the manufacture of metal products and own the mines upon which they depend for their raw material, a policy of rapidly depleting the higher-grade ores with the loss of lower-grade and marginal ores would shorten the life of the enterprise as a whole or force it to enter the market for ores from outside sources. The sacrifice of ore reserves to maximum profits from the mine would therefore be a detriment to the company and actually lower the value of its assets.

Since the question of conservation is assuming more importance and public attention is being focused upon it, the trend of legislation is likely to be toward measures that will prevent wasteful operation of mines for the sake of large immediate profits.

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[4] Hotchkiss, W. O., and Parks, R. D., Work cited in footnote 3 (p. 482).

PROPERTIES WITH LITTLE OR NO ASSURED ORE RESERVES

In the case of properties having large assured ore reserves and long life, the effect upon present worth of additional ore that may be found later is relatively small because the profit to be derived from it is remote.

In the case of a property with small proved reserves, however, the present value of profits to be derived from the known ore may be only a small fraction of the actual value. The value of the known ore may be computed readily by the regular methods, but evaluation of the potentialities of the property requires the exercise of judgment based upon a knowledge of the ore habits in the district and of structural and other geological controls. The question of probable and possible ore has been discussed in the section on Estimation of Ore Reserves, where the uncertainties regarding conclusions as to tonnage and grade were pointed out. As a basis for valuing this unproved ore, the usual calculations may be employed, using, however, a high rate of return because of the uncertainties involved. It is apparent that widely different results may be derived by different engineers, depending upon their relative optimism or pessimism, which will be reflected both in the tonnages and grades of ore estimated and in the percentage of return judged to be commensurate with the risk to the investment. Likewise, the price that is paid for such a property will depend largely upon the confidence of the purchaser in its possibilities and the size of the risk his speculative instinct prompts him to take on the chance of large profits. Often the risk may be minimized by obtaining an option to purchase that permits a period of development before substantial payments on the purchase price must be made.

UNDEVELOPED PROPERTIES

In the valuation of undeveloped, unequipped properties, whether having large or small ore reserves or none at all, consideration must be given to time and expenditure required to equip, develop, and bring the mine into profitable production. The expenditures up to the time of profitable production (including the purchase price of the property), with interest thereon, must be amortized from profits. The tables referred to above include, as stated, factors for computing present worth on profits deferred for from 1 to 4 years.

Rates of progress and costs of the various phases of mine development are engineering matters that were discussed in preceding sections of this bulletin.

CONCLUSION

The discussions, illustrations, and data brought together in this bulletin, largely from circulars published in nonpermanent mimeographed form, are based on information gathered and studies made during 1928 to 1938, inclusive.

At the beginning of this period iron and nonferrous base-metal mines were in general working at or near capacity, whereas gold mining had been declining gradually for more than a decade. However, during the last 5 or 6 years the former have been passing through a period of low metal prices, high costs, and restricted markets that has entailed curtailment of production and even complete suspension of operations for considerable periods. Ingenuity, resourcefulness, and engineering skill have been summoned to meet these conditions, and the development of new equipment and cheaper methods of working has resulted in the introduction of improved practices at many of the mines and mills discussed herein since the original studies and data were gathered.

Although the authors have endeavored to keep in touch with the changes at these properties and have noted certain new developments in the text, many of the cost and other data apply to the earlier practices. In some instances these data would have to be revised considerably to apply to present conditions, but in many others the changes would be only nominal.

The same period has witnessed a revival of gold mining, which was given impetus in 1934 by an increase in the price of gold from $20.67 to $35 per ounce. Most of the data on gold mining were gathered during 1934 to 1938, inclusive.

Mining is a business requiring for its successful conduct the exercise of a high degree of engineering, managerial, and executive skill. Mineral deposits are exhaustible and every operating mine is therefore a wasting asset. Mining possesses inherent risks peculiar to it but not necessarily greater than those of many other forms of business. The great majority of ordinary business enterprises is subject to widely varying degrees of prosperity and adversity, and even the most successful usually have but a few years of large earnings interspersed with many lean years. From the viewpoint of safety of long-term investment, mining enterprises built upon large reserves of ore have much in their favor; these reserves constitute a supply of valuable basic raw material that always has intrinsic value and that, under sound management over a period of time, can be made to yield adequate returns on the capital required for their exploitation.

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